Metallurgical and Chemical Engineering

A Consolidation of ELECTROCHEMICAL AND METALLURGICAL INDUSTRY and IRON AND STEEL MAGAZINE.

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The Eighth International Congress of Applied Chemistry.

The Eighth International Congress of Applied Chemistry has passed and gone into history, never to be forgotten, we are sure. What an enormous undertaking it was a few figures will show.

There were 4500 members, of which 2173, coming from thirty different countries, attended the Congress and registered, including 254 ladies. Seven hundred and fifty papers were presented, of which 570 were in print before the Congress assembled. Besides the four general lectures by official representatives of different countries there were 117 single section meetings, with an average attendance of 34 per sectional meeting, and 32 joint section meetings, with an average attendance of 183 per meeting. If we leave out of consideration the joint sessions an average attendance of 34 per sectional meeting means that at any time more than 800 out of the 2173 members who had registered were present at one or the other of the meetings of the 24 sections.

To appreciate the meaning of these figures we need a standard of comparison. The International Electrical Congress, held in 1904 in St. Louis, seems particularly appropriate for this purpose, since it is recognized as perhaps the greatest success of international scientific and engineering congresses held in this country in the past. The number of members attending the Electrical Congress was 700 (against 2173 at the Congress of Applied Chemistry). About 160 papers were read (against 750). There were eight sections (against twenty-four). The ratio being 3 to 1 in the numbers of attending members and sections and 5 to 1 in the number of papers in favor of the Congress of Applied Chemistry, its huge magnitude is clearly established.

The enormous scope of the Congress is best indicated by the special fields to which the work of the twenty-four sections was devoted: Analytical chemistry; inorganic chemistry; metallurgy and mining; explosives; silicate industries; organic chemistry; coal tar colors and dyestuffs; industry and chemistry of sugar; india rubber and other plastics; fuels and asphalt; fats, fatty oils and soaps; paints, drying oils and varnishes; starch, cellulose and paper; fermentation; agricultural chemistry; hygiene; pharmaceutical chemistry; bromatology; biochemistry, including pharmacology; photochemistry; electrochemistry; physical chemistry; law and legislation affecting chemical industry; political economy and conservation of natural resources. Clearly every field of human endeavor in which the work of the chemist counts was represented.

The lasting monument is naturally the set of the Transactions of the Congress. Twenty-four volumes were available in print when the Congress opened, which is an achievement which is without parallel, we believe, in the history of international congresses. That it was appreciated is evidenced by the fact that 1313 members withdrew their sets before the Congress was over. It may further be truly said that the

papers in their perplexing variety were throughout of a high character. That chemistry has not yet lost its revolutionary character and that chemical invention is still able to change industries and rearrange the geography of the world was illustrated by such papers as those by Dr. Eyde on the Norwegian industry of fixation of atmospheric nitrogen, by Dr. Bernthsen on the synthesis of ammonia, by Dr. Duisberg and Dr. Perkin on the synthesis of rubber.

The social functions were delightful and every member enjoyed immensely and to the utmost the personal intercourse with so many colleagues and leaders from so many countries. This is, of course, the main reason why people attend an international congress. They can read any paper with greater leisure and convenience later on at home, but only at the Congress are they able to see what kind of a man has written it. The simple privilege of every chemist who attended the Congress to see, hear, and meet congenial men from many countries, to discuss questions of mutual interest with the great leaders in his chosen field—this privilege must have been a source of inspiration the value of which is inestimable. But we think that the uplifting effects of the late Congress will be felt far beyond chemistry. This intermingling of men, coming from many countries, speaking different languages, interested in many different lines of work, must have led to an interchange of ideas on those broad problems in which the citizens of all countries are equally interested as brothers and citizens of the world. We sincerely hope that the late Congress has done more towards friendship and peace between nations than any peace society or any number of battleships can ever do.

Since our whole issue of September 12th and part of the present issue are devoted to reports of the Congress and to abstracts of papers, there is no necessity to go any more into details at this place. But a tribute of appreciation is due here to all who contributed to the success of the Congress, that is, to all its members who joined in serious work and in happy enjoyment of the occasion, to all the members of committees who spent uncounted hours and days of hard and yet pleasant work in the preparation of the Congress, and to its four chief officers: the president, Dr. Wm. H. Nichols, the honorary president, Dr. Edward W. Morley, the treasurer, Mr. W. J. Matheson, and the secretary, Dr. Bernhard C. Hesse. Through their indefatigable energy and the combined efforts of all, the Eighth International Congress of Applied Chemistry was made a success of which American chemistry and this whole country may well be proud.

Advances in the Metallurgy of Aluminium.

That the laws of supply and demand govern chemical and metallurgical invention as well as any other field of human endeavor, is well illustrated by the present activity in the metalurgy of aluminium. The demand for aluminium is increasing enormously. Dr. Heroult remarked last year in his delightful little speech on the occasion of the award of the Perkin medal to Dr. Hall: "Aluminium is wedging its way in the metal market. It looks probable that in a course of ten or fifteen years the consumption of the new metal will be equal to that of copper and that after the golden age, the stone age, the bronze age, and the iron age, we may have the aluminium age"

The demand is clearly not only for more, but for cheaper aluminium, and chemical invention will supply it.

Besides a new alumina process of a distinguished American chemist, which is now about to be tried out on a sufficiently large scale, but is being carefully kept secret for the present, the Serpek process, in use on a large experimental scale in a plant in France, looms prominent. As is well known, with aluminium nitride as an intermediate product, it yields pure alumina and ammonium sulphate. It is, therefore, technically an interesting combination of the production of pure alumina for aluminium manufacture with the fixation of atmospheric nitrogen. Commercially it may be said that the Serpek process is now in a decidedly "interesting condition."

And now, on the last day of the International Congress of Applied Chemistry, Mr. Alfred H. Cowles, the distinguished pioneer of electrochemistry who needs no further introduction to our readers, announced for the first time in public the principles of the new Kayser-Cowles process which is to be operated in a plant at Sewaren, N. J. It is an ingenious and ambitious process yielding aluminium, hydrochloric acid, caustic alkali, and white hydraulic cement from clay, salt and lime. The process certainly appears sound chemically, and it may be hoped that any chemical engineering difficulties which may turn up, as they always do in the introduction of a new process into operation on a large scale, will be overcome as a matter of course. Mr. Cowles' most interesting paper is published, practically in full, in this issue.

Altogether the metallurgy of aluminium is at present full of promise of almost revolutionary developments and deserves close watching.

Methods of Sampling and Analysis.

There is something to be said both for and against the policy of "standardizing" methods of technical procedure, such as the sampling and analysis of ores and metallurgical products. The opponents of the idea usually contend that the very act of adopting a standard method is a bar to progress, and removes the incentive to research for better methods; that a standard method once promulgated as such by some authority is likely to gain a strong position, and be retained in spite of possible deficiencies. However much opinions may differ on this subject, we believe that even the opponents of the policy must recognize the fact that the prevailing industrial conditions of the day warrant an attempt to have the best methods uniformly adopted, with no implication that these may not be changed from time to time.

In fact it may be said that the present need for uniform methods is the result of research, and that when a method is proposed for uniform adoption it represents the best that has been evolved from the experience of many competent men. Instances could be multiplied to show the impossibility of two analysts producing concordant results on the same material without following practically the same methods of sampling and analysis. Hence the commercial need for uniformity. The demand becomes more insistent when a large company employs a number of chemists situated at different places, and all working on the same materials. It is scarcely less imperative among individual companies buying each other's products. Business is expedited because reliance can be placed in accurate results.

Elsewhere in this issue we publish the methods of the United States Steel Corporation for the commercial sampling and analysis of pig iron, prepared by the chemists' committee of that corporation and presented in form of a paper before the Eighth International Congress of Applied Chemistry. This is the third of the excellent reports which have emanated from the same source. The first, which we published in February, 1909, dealt with the commercial sampling and analysis of iron ores. The second, which appeared in our issues for June and July, 1911, outlined methods for the technical sampling and analysis of gases. They resulted from a condition already mentioned, namely, that prior to their adoption a number of men were carrying on the sampling and analysis of these materials under varying conditions with consequent non-conformity in results. By careful consideration of the methods used in each laboratory under the corporation's control, general methods were evolved which undoubtedly are preferable to some that were used before. In this way the standard of all the work has been raised.

In addition to the reports already published by the Chemists' Committee of the Steel Corporation, methods of sampling and analysing steel, ferro-alloys, etc., are being investigated and will be issued in due time, when we hope to publish them in this journal. The complete series will form a most valuable critical compilation of useful methods based on extended research and wide experience.

The Steel Production Statistics.

The American Iron and Steel Institute reports production of steel ingots and castings and of rolled iron and steel in the first half of 1912 as follows, the figures representing gross tons:

Basic open-hearth steel	8,927,579
Acid open-hearth steel	514,127
Bessemer steel	4,868,306
Crucible steel	53,476
Electric steel	6,882
Miscellaneous steel	3,331
Total steel	14,373,701
Steel castings, included above	490,218
Rolled iron	779,481
Rolled steel	10,422,338
Total rolled material	11,201,819

The Institute succeeds the American Iron and Steel Association in the gathering of American production statistics. Under the management of Mr. James M. Swank, the Association has gathered these statistics for forty years, Mr. Swank's first work having been the gathering of 1872 statistics, early in 1873. The Association has gathered pig iron production statistics semiannually, and all other statistics annully, and its work closes with the presentation of full statistics for the year 1911, and of pig iron statistics for the first half of the present year. The Institute at once esablishes a new practice, of gathering steel statistics for semi-annual periods, and for this departure it is much to be commended. A curious fact is that the statistics for several items are presented for the first half of 1912 when the Association has not yet published the corresponding statis-

tics for 1911, and one is curious to know how it is that the Institute shows more promptness in its maiden effort than the Association does after forty years of experience. The Association's figures have established such a reputation for accuracy, being admittedly much better in point of accuracy and completeness than those of any other iron and steel producing country, and it may perhaps not be amiss to suggest that the trade would welcome a positive assurance from the Institute management that the Association's standard of accuracy has been maintained.

The production of steel in the first half of this year cannot be compared with previous records, because half-yearly statistics of steel production have not hitherto been gathered. It is well known, however, that the most active period in the past was the first half of 1910, and if that half year produced the same proportion of the year's total steel production as obtained in the case of pig iron—for which semi-annual statistics are available—the output was approximately 14,400,000 tons of steel ingots and castings, substantially the same quantity as now reported for the first half of this year. A new record has not been made, therefore, at least by any considerable margin. The present half year, of course, is easily making a new record, as production is at a much greater rate than in the first half of the year.

A new tonnage record for basic open-hearth steel was made in the first half of this year, for the basic open-hearth steel production was 62.1 per cent of the total steel output, and exceeded that of Bessemer steel by 84 per cent, whereas in 1910 basic open-hearth steel was only 58.7 per cent of total steel, and exceeded Bessemer steel by only 64 per cent. The decadence of the Bessemer steel process, not merely relatively, but also absolutely, is further assured, for the record tonnage year in Bessemer steel was as far back as 1906, with 12,275,830 tons, and the production in the first half of this year was only 4,868,306 tons. No new Bessemer steel plant has been built since that of the Youngstown Sheet & Tube Company, which made its first steel in the summer of 1906, whereas two very important Bessemer departments, those of the Carnegie Steel Company at Homestead and Duquesne respectively, have not made Bessemer steel since 1907.

The production of steel ingots, disregarding castings, in the first half of 1912, is reported at 13,883,483 tons, while the production of rolled steel, in the form of rails, wire rods, plates, shapes, sheets, merchant mill products, skelp, etc., is given at 10,422,338 tons, representing an apparent loss of 25 per cent. This is slightly in excess of the average apparent loss in recent years, and suggests the possibility that in some cases the mills have grown more liberal in cropping, when in most cases the basic open-hearth steel furnace stands ready to remelt the material. In discussions of the progress of the steel industry in the matter of tonnage production, it is customary to refer to these statistics of the production of steel ingots and castings as representing "the production of steel." In one sense this is steel production, but the more accurate viewpoint is to regard the production of rolled steel, which is actual commercial material. A large part of the ingot material merely passes over the ingot scales, and passes not into the steel market, but into the open-hearth steel furnace, to be melted and pass a second time over the ingot scale.

Old Freibergers in America.

A very enjoyable and enthusiastic meeting of old students of the Freiberg Bergakademie was held at the Hofbräu Haus, Broadway and Thirtieth Street, New York City, on Friday, September 13.

At this meeting it was unanimously decided to arrange for a dinner in New York City near the Christmas holidays and at that time form a permanent organization.

Among those present at the first meeting were: Dr. F. Heberlein, of Frankfort, a/M., and Prof. R. Hoffmann, Professor of Metallurgy at the Clausthal School of Mines, Clausthal, Germany. Both of these gentlemen are well known not only as old Freibergers but as highly distinguished metallurgists.

The organizers are having some trouble in getting correct addresses, since mining and metallurgical men are a wandering crowd.

All old Freibergers are, therefore, asked to send in their names and addresses, with others they may know of, to Mr. C. L. Bryden, 1015 Myrtle Street, Scranton, Pa., so that a notice of the next meeting can be mailed to them.

The Iron and Steel Market.

In specifications for finished steel products filed with the mills September has shown no improvement over August, as normally it should on account of the season, but the heavy specifying in July and August was altogether abnormal. September has fully maintained the pace of August, showing specifications in excess of current shipments, so that the mills have a somewhat larger volume of actual specifications on their books than at the close of August and a much larger volume than at the close of July. As tonnage taken at lower prices is being constantly worked out, the average realized price on shipments is constantly increasing, giving the mills better profits, although profits are still relatively small.

Production of steel continues to be at the maximum capacity of mills, and is somewhat improved over the summer months on account of weather conditions. The labor supply is no better, and hampers operations in many minor ways. The car supply is also a serious problem, and while the current make is being shipped, the shipments are maintained only by the exercise of constant vigilance, and conditions are certain to be worse in October and November on account of the drain for moving the crops.

It has become the regular practice with many mills to make minor repairs on box cars at their own expense, to render them weatherproof.

In this way many cars have been made available which otherwise could not be used.

Early in September the independent sheet and tin plate mills began advancing prices, and the leading interest concurred, establishing black and galvanized sheets and tin plate on a higher basis, black sheets being advanced from 2.05 to 2.15 cents, galvanized sheets from 3.15 to 3.30 cents and tin plates from $3.50 to $3.60. The increased spread between black and galvanized sheets is to be noted, 1.15 cents per pound on No. 28 gage.

This increase was due to spelter having a record high price, about 7½ cents, East St. Louis. As large a spread between black and galvanized has sometimes prevailed in the past, with lower spelter prices, and the present spread is not regarded as sufficient.

During the month several mills announced advances of $1 a ton in bars, plates and shapes, applicable to first quarter business, and making the advanced prices 1.35 cents for bars and 1.45 cents for plates and shapes. In harmony with a policy recently adopted, of curbing the advancing tendency in the steel market, the United States Steel Corporation did not follow this lead, but is selling these products in a limited way for next year at the old prices. On account of the mills being so well sold up there is room for a variety of prices in the market.

While the selling of bars, plates and shapes to ordinary buyers for the first quarter has not been large, the market has been definitely opened in sheets and tin plates. The leading interest formally opened its books on Monday, Sept. 23, for sheet orders from jobbers and ordinary manufacturing consumers, and at the same time for tin plate orders from ordinary buyers for first quarter, but from makers of packers' cans for the regular season. The opening is an early one in both sheets and tin plates, but was practically forced by the peculiar conditions, the mills being practically sold up for this year. In both sheets and tin plates they have actual specifications for about half the prospective make in the remainder of the year, and as specifications are coming in very freely on unexpired contracts they are certain of full operations throughout the year without the booking of additional contracts.

The finished steel market is thus gradually opening for the new year, and with pressure on the mills for deliveries even heavier than it was there is less occasion to fear that the advent of the new year, with the expiration of many low-priced contracts, would see a lull in buying activity. While buyers will have to pay materially higher prices for next year's materials, the momentum acquired will probably carry the market into the new year without a jar.

Pig Iron.

Pig iron prices in all districts have been advancing almost spectacularly, considering the slowness with which they advanced for many months when conditions in the steel trade were all extremely favorable. The somewhat sudden change in the aspect was due to several causes. In the first place, large stocks of merchant pig iron had accumulated in 1910, after the ill-advised boom in 1909, and these were not well liquidated until about the middle of this year. In the second place, the Connellsville coke operators became seized last May with the idea that they should secure a large advance in coke, about $1 a ton, even though prices for pig iron obtained by their customers did not justify such an advance. They held that a coke advance should force a pig iron advance, which is not the customary procedure. The furnaces, as a rule, were disposed to hold pig iron prices down until the coke operators were willing to accept lower prices. The deadlock was broken late in July, operators accepting $2.25 and possibly less when they had been holding out for $2.50. Under strictly open market conditions, however, the coke market has since advanced to $2.50 or higher. Finally, the merchant furnaces in operation, being quite well sold up, were averse to seeing a paper advance which would benefit them little and would presumably attract many idle furnaces into the active list. The position is now worked out. The stocks are disposed of, coke is selling under open market conditions at a high price, and furnaces are ready to sell for next year, while there is no longer hope of discouraging idle furnaces from blowing in. Pig iron in the past month has advanced sharply in all districts, with fairly heavy selling for delivery in the first quarter of the new year. We quote: Bessemer, $16; basic, $15; No. 2 foundry, $14.75@$15, f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh: No. 2X, delivered Philadelphia, $16.50; No. 2 foundry, delivered Chicago, $17; No. 2 foundry, Birmingham, $13 to $13.50.

Steel.

Unfinished steel has become extremely scarce for any delivery this year, and mills are not naming prices for next year. As representing nominally the general level, we quote, f.o.b. maker's mill, Pittsburgh or Youngstown: Bessemer billets, $23.50 to $24; sheet bars, $24; open-hearth billets, $24.50 to $25; sheet bars, $25.50; rods, $26.50 to $27.

Finished Steel.

Current prices, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.40 to 1.45 cents.

Shapes, 1.40 to 1.45 cents.

Steel bars, 1.30 to 1.35 cents.

Common iron bars, 1.50 cents, Pittsburgh; 1.45 to 1.50 cents, Chicago; 1.45 cents, Philadelphia.

Steel hoops, 1.50 cents.

Sheets, blue annealed, 10 gage, 1.55 to 1.60 cents; black, 28 gage, 2.15 cents; galvanized, 28 gage, 3.30 cents; galvanized corrugated, 3.35 cents.

Tin plates, $3.60 for 100-lb. cokes.

Merchant steel pipe, 79 per cent off list.

Steel boiler tubes, 3½ to 4½ in., 73 per cent off list.

Standard railroad spikes, 1.80 cents.

The Western Metallurgical Field.

Concentrating Low-Grade Silver-Lead Ore at Aspen, Col.

One of the most difficult tasks in profitably concentrating low-grade ore is that undertaken by the Smuggler Leasing Company, in an effort to revive the famous old mining district
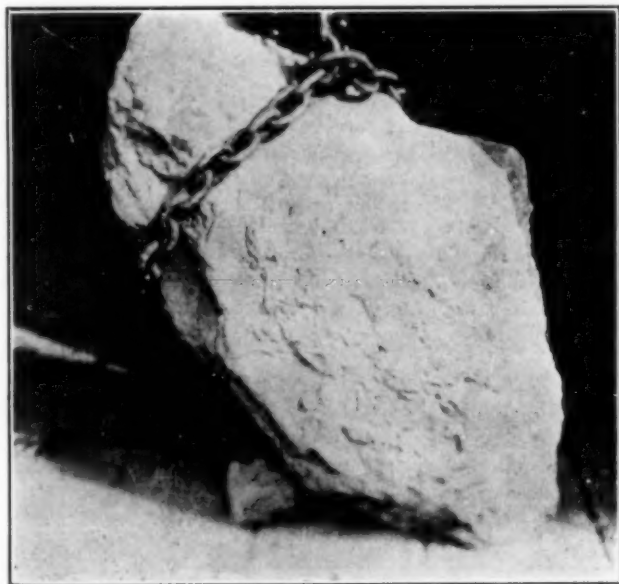

FIG. 1.—LARGE SILVER NUGGET FROM SMUGGLER MINE, ASPEN, COL.

of Aspen, Col. Prior to 1893-4 Aspen was one of the remarkable silver producers of the West. The Smuggler, Mollie Gibson, Aspen and Durant mines were well known for their production of native silver and high-grade silver-bearing galena. Silver nuggets of great size and value were not of uncommon occurrence, and rich pockets of ore were opened frequently. In the accompanying photograph is shown the largest silver nugget known, which was taken from the Smuggler mine. It weighed 1840 lb. and was of a fineness equal to that of sterling silver.

Conditions at Aspen have changed, however, since the bonanza days. The price of silver is greatly reduced; the grade of the remaining ore is very low; the expense of pumping and timbering is heavy; and altogether the situation demands the most careful management. A large tonnage of ore must be handled and prospecting for new deposits must go hand in hand with production. It is worthy of mention that those who are responsible for the operations of the Smuggler Leasing Company are men who formerly realized great wealth from the

old mines, and who still believe that there is more money to be made by properly treating the low-grade ore. There is also the ever-present hope that bodies of rich ore may be opened in the course of development.

The Smuggler mill is an old structure of German design, and still contains some of the German machinery, notably the jigs. As originally designed, a great deal of manual labor was required in shoveling and tramming concentrates, and in elevating the crushed crude ore on platform elevators. These features have been displaced, of course, by suitable launders and elevators. The original capacity was 125 tons per day, but by remodeling this has been increased to 350 tons.

The concentrating ore is argentiferous galena in a gangue of limestone and quartz. The mineral is fine-grained and requires regrinding to liberate the smallest crystals, thus creating a large amount of slime. Further, the gangue is soft and slimes badly in crushing, and as the lime carries silver the recovery of this metal is made increasingly difficult. By a local custom the grade of the ore is stated in "points," each per cent of lead and each ounce of silver per ton being a "point." Thus, "12-point" ore contains, say 10 per cent lead and 2 ounces silver per ton. The average grade of ore being milled at this time is about as given. The recovery of lead will average about 85 per cent, but silver loss is high and only 60 per cent of the silver is recovered.

It is the aim of the concentrating process to enrich this ore to an average grade of at least 15 to 16 per cent lead and 4 to 5 oz. silver per ton, or in the local vernacular, 20 points, as this is the lower limit of profitable grade of concentrate. Frequently the grade will average 18 per cent lead and 5 oz. silver per ton for a period of a month. This, of course, represents mixed coarse and fine concentrates, the former being higher grade and the latter lower than the average.

It is estimated that the company can treat its low-grade ore at good profit if the mill handles 350 tons per day of ore containing 10 per cent lead and 2 oz. silver per ton. But the capacity of the regrinding division is not equal to that of the coarse concentrating department, hence at present the latter is run only two shifts per day and the former three shifts.

Following is an outline of the concentrating system. Bottom-dump railroad cars convey the ore from the mine to the mill and discharge into a hopper beneath the tracks. From here the ore is conveyed to the crushing department by an inclined link-belt conveyor. A shaking feeder, punched with 1-in. holes, delivers the oversize to the crusher. Concentration begins at 16 mm. in a Harz jig, where about 8 per cent of the original mill feed is removed. Jigging is continued on the following sizes, 12, 8, 6, 4½ and 3 mm., and on the two coarse products from hydraulic classification of the undersize of the last (3 mm.) screen. Three Wilfley tables treat the balance of the classified material, completing the coarse concentration.

In the regrinding division the middlings from the jigs are ground to 20-mesh in Huntington mills. Ton-cap and rek-tang screens are being tested in these mills. Wilfley tables, smooth and corrugated belt vanners are used here, the concentrates being conveyed to central elevators by shaking launders. An extensive system of settling boxes is provided to recover mineral slime which flows away from the various machines. The material thus recovered contains from 15 to 18 per cent lead and is mixed with the coarser concentrates.

Mention has been made of the old jigs originally installed and still in use. These have a crank-arm mechanism to control the plunger stroke, which is seldom used in modern jigs. The object of the mechanism is to impart a quick pulsion and slow suction to the bed of ore on the screen, so that after each downward stroke of the plunger the ore bed would be free to settle in nearly still water. The expense of maintaining these jigs has been found excessive, especially where the old construction of a sectional shaft is still used.

Mr. Walter W. Davis, of Leadville, is manager for the Smuggler Leasing Company, being represented locally by Mr.

Guy Newton. Milling operations are under the direction of Mr. W. H. Cornwall.

Company Reports

The report of the **Ray Consolidated Copper Company** for the second quarter of 1912 shows that the total amount of ore treated for the second quarter was 374,609 tons, as compared with 301,674 tons. The average grade of the ore for the quarter was 1.72 per cent copper, almost exactly the same as for the first quarter. The average recovery for the quarter was 69.37 per cent; the average for the previous quarter was 68.85 per cent. The recovery in pounds, per ton of ore, was 23.9 as compared with 23.6 lb. for the preceding quarter. The net tonnage and average grade of concentrates produced was 22,-879.51 tons containing 19.563 per cent copper, as against 17,637 tons for the first quarter containing an average of 20.19 per cent copper. The average cost per pound of net copper produced for the quarter, after making allowances for the smelter deductions, was 9.954 cents. This cost was 10.19 cents for the preceding quarter.

In comparing the cost for the second quarter with those of the first, it should be noted that during the second quarter slightly over half a cent per pound of copper produced was charged to create a fund to extinguish development expense, which charge was not made in the first quarter.

The improvement in operating and production costs, as tonnage is increased, is well indicated by the fact that during the month of June the cost per pound of copper derived, as above stated, was 8.95 cents per pound.

During the quarter, five sections of the Hayden plant were available for continuous operation, but for a large portion of the time it was only necessary to use four of them to accommodate the tonnage produced at the mines. Toward the end of the quarter the five sections were in continuous operation. The sixth section has been finished and is ready for use as required. Construction work on the seventh section is progressing at such a rate as will insure its completion when needed which will probably not be before the first of October. The cost of milling for the quarter, including a proper apportionment of all general and fixed charges and expense of every nature, was 45.52 cents per ton, as compared with 56.07 cents for the first quarter. The mill, power plant, water system and all other accessories are operating in a thoroughly systematic and satisfactory manner.

The smelter of the American Smelting and Refining Company, at Hayden, has been in continuous operation since early in May, handling concentrates as produced with one reverberatory furnace. The concentrates are proving admirably adapted to direct smelting without admixture of other ores or fluxes, except the usual quantity of lime, and are therefore taking the base smelting charge without any penalties whatever for refractory qualities.

Steady improvement is shown in the operations of the **Chino Copper Company,** as published in the report for the second quarter of 1912.

The total amount of ore treated for the quarter was 167,025 tons as compared with 160,400 tons for the preceding quarter. The average grade of the ore treated for the period was 2.04 per cent copper. The production of concentrates amounted to 9035 tons, containing 23.74 per cent copper, as compared with 8048 tons of concentrates containing an average of 20.33 per cent copper for the preceding quarter. The average recovery for the quarter was 62.96 per cent, corresponding to 23.74 lb. of copper per ton of ore treated, as compared with a recovery of 20.4 lb. for the preceding quarter.

In our last report it was explained that the low recoveries were due to the large amount of oxidized copper minerals existing in the ores then being mined. This condition continued to a considerable extent throughout the first two months of the quarter under discussion. By the first of June, however, a better quality of ore was available as deeper levels in the steam shovel pits were reached, and for the month of June the

average recovery amounted to over 71 per cent of the copper contained in the ores milled.

The average cost per pound of copper produced during the quarter after deduction of smelter allowances, but without crediting miscellaneous income, was 8.02 cents as compared with 7.68 cents for the preceding quarter. The actual cost per pound of direct operations was substantially lower for the second quarter than for the first, but at the beginning of the second quarter a charge was established of 30 cents per ton of ore mined to create a fund for the retirement of development and stripping expense. This charge amounted for the quarter to over 1.3 cents per pound of net copper produced, and the direct operating cost of production for the second quarter was accordingly about 1 cent a pound less than for the first quarter.

Two sections of the plant at Hurley were in continuous operation throughout the quarter, barring ordinary delays. The third section was ready for operation about the first of April, but for various reasons, only two sections, as above stated, could be operated continuously. During the month of April and part of May the water supply was insufficient to operate more than two sections on account of delays in completing the permanent water system. During the month of May some difficulty and delay were experienced in the fine crushing department in treating the first ores mined from the Hearst section. This was due to their clayey nature, which resulted in their retaining a very high percentage of moisture, rendering fine screening difficult and necessitating some changes in the screening plant in the fine crushing department.

During the month of June temporary labor disturbances at the mine were responsible for some shortage in ore supply, and for this reason it was not until after the close of the quarter that three sections could be operated regularly.

Final results for July operations are not available at the time of writing this report, but for that month an average of over 3000 tons daily has been treated with two sections running continuously and a third only a part of the time. The grade of ore has increased somewhat as compared with the second quarter's results and the estimated extraction for the month is above 72 per cent, which is excellent in view of the fact that the ores still contain a considerable quantity of oxidized copper minerals difficult of recovery. The grade of concentrates is increasing with the improvement in quality of the ore. For the month of June the average was about 26 per cent and for July the concentrates will contain over 30 per cent copper. This is an important factor in the reduction of per pound costs for transportation and smelting of concentrates. The production for July, with three sections only in operation, will be more than double that of any preceding month, or well above 3,000,-000 lb.

The fourth section of the mill was practically completed at the time of writing this report, and was put into commission about the 15th of August, and it is not probable that the four sections then in commission can be operated at their full tonnage until late in September. The fifth and last section will be completed about the end of September and it is expected that mining and transportation capacities and facilities will have reached a stage where the estimated capacity of 5000 tons per day for the plant can be supplied in October. Under favorable conditions the concentrator has developed a capacity of nearly 1500 tons per section and it is certain, therefore, that the completed plant will be able to mill considerably in excess of 5000 tons per day for which it was designed.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in May, 1912, was 61. The total quantity of ore milled during that period was 2,227,532 tons. There were 10,073 stamps in operation with an average duty of 8.21 tons per 24 hours. Tube mills in commission numbered 266. The yield for the month was 779,662 fine ounces gold.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

Peat as a Source of Power.

At the summer conference of the Institution of Mechanical Engineers at Belfast, Mr. **H. V. Pegg** presented a paper on the "Commercial Utilization of Peat for Power Purposes." The author arrived at the conclusion that, in view of the generally poor results obtained by means of costly plant for preparing peat, the employment of peat on a commercial basis depended on its being used as nearly as possible in the state in which it was cut from the bog lands. Preliminary experiments, some seven years back, with air-dried peat in a special gas producer, showed that the extraction of the tar from the gas was preferable to attempting to make gas of it, and that the producer must be comparatively unaffected by the amount of moisture in the peat.

About two years ago some tests with experimental plant at the works of Messrs. Crossley Bros., Openshaw, resulted in that firm making a special producer plant with which, in October, 1911, a six hours test run at a load of 250 B.h.p., using peat containing 18.98 per cent of moisture, gave an average consumption of 2.55 lb. of peat per B.h.p. hour. The peat used during November and December was very wet; its moisture varied very much; and, although not exactly determined, was estimated at quite 70 per cent; yet the plant worked quite as well with this as with the drier peat, but of course the consumption was proportionately greater.

For the removal of tar from the gas—which was the chief difficulty—an ample water spray was found superior to coke scrubbers as these soon became clogged. Most of the tar was thrown out by a centrifugal extractor; but unless the gas were sufficiently washed and cooled by plentiful water spraying some tar passed by the extractor, deposited in the gas mains, and ultimately entered the gas engines.

The plant now in use was capable of being run for three weeks without being cleaned, although as a matter of precaution it was usually cleansed weekly; and the amount of wash water had been increased to 7 gal. per B.h.p. per hour. The tar recovered was about 5 per cent of the weight of fuel used, and at first some of the tar was sold at 35/- per ton; but its persistent pyroligneous odor, which also clung to all the oils distilled from it, stopped the sale. The tar contained up to 50 per cent of water and much solid matter.

At present the whole of the factory at Portadown was being run on peat fuel exclusively with a total consumption of about 44 tons per week, about half of this amount being used in the producer plant; but the consumption varied with the kind of peat from 35 tons of good black fuel to 54 tons for light surface peat. From figures furnished by the works manager, the cost of the peat fuel showed a saving of £15-5-6 per week after allowing 15/- per week for extra labor, thus: 8½ tons of anthracite at 35/- and 19 tons steam coal at 17/ = £31-0-6; against 50 tons of peat at 6/- = £15-0-0.

French Aluminium.

An association of French manufacturers under the title of "L'Aluminium Français" has secured the patent rights for all the world of the Serpek process of manufacturing aluminium: and, according to a British Consular report from Lyons, is proceeding to erect extensive works with a 40,000-hp equipment. It is proposed to put down the rolling plant near Chambéry, in the expectation that the ample water power obtainable will result in very considerable saving in working costs.

The association acquired the patent rights from the Société des Nitrures, who in turn derived them from the Swiss holders and conducted manufacturing operations on an experimental scale in the neighborhood of St. Jean de Maurienne. The rights for the United States have been disposed of to a subsidiary company (which is erecting a plant in North Carolina, as noticed in our July issue, page 445), but at present there is not any mention of the process being taken up by British manufacturers.

Electrolytic Hypochlorite.

The medical officer of health for the Borough of Poplar, Dr. **F. W. Alexander,** has recently issued his annual report for 1911, which gives particulars of the production and employment of hypochlorite during the last six years. Since the plant was first put down about 200,000 gal. of solution, containing an average of 5 gr. of available chlorine per litre, have been produced at a total cost for materials of rather less than £380; and the output for 1911 was 53,063 gal. at a cost of £108-9-0 for materials and electricity.

The use of the fluid in the public baths of the borough has been continued at the rate of one part of chlorine to 2,000,000 parts of water, and the bath superintendents report that this addition is perfectly effective in preventing formation of slime in the water, that no slime is found at the bottom, nor is there any offensive smell when the baths are emptied for cleaning.

In the Public Health Department the fluid has for some years entirely displaced carbolic disinfectants, and it is also exclusively used at the institutions of the managers of the sick asylums and of the Board of Guardians. The number of rooms disinfected during the year was 1626, and the articles of bedding, clothing, etc., totalled 41,143. Dr. Alexander deals with the contention that hypochlorite could be produced more cheaply by lixiviation of bleaching powder, by pointing out that the price of this substance fluctuates considerably, that the powder deteriorates rapidly on exposure to air, that the lime residues would be a nuisance, and that the cost of supervision and labor would be increased; and he points out that it is much easier to work out figures on paper than to practically produce results.

The fluid made at Poplar is termed "hypochlorite of magnesia," but the report shows that the materials used for the last year's working include over 8 tons of sodium chloride, and a ton or more of caustic soda, in addition to about 5¾ tons of magnesium chloride.

The Iron and Steel Institute.

The autumn meeting of the Iron and Steel Institute will be held at Leeds on September 30, and the four following days. Among the papers submitted are the following: "On Nitrogen and Iron," by J. H. Andrew; "On the Solubility of Cementite in Hardenite," by Dr. Arnold and L. Aitchison; "On the Solubility or Diffusion of Hardenite in Ferrite," by Dr. Arnold and C. Chappell; "On the Gases Evolved on Heating Steel to Its Melting Point in a Vacuum," by G. Wesley Austin; "On Allotropy in General, and That of Iron in Particular," by Dr. C. Benedicks; "On a new Type and Method of Construction of Large Gas Engines," by A. E. L. Chorlton; "On the Thermal-magnetic Transformations of 25 per cent Nickel Steel," by Dr. Colver-Glauert and Dr. Hilpert; "On a Method of Producing Sound Ingots," and "On a New Method of Revealing Segregation in Steel Ingots," by Sir Robert Hadfield; "On the Magnetic Properties of Manganese and Nickel Steels," by Dr. Hilpert and Dr. Mathesius; "On Steel Works Yields," by P. Longmuir and W. H. Robinson; "On Rolling Mill Practice in the United States," by Dr. J. Puppe, and "On the Growth of Cast Irons after Repeated Heatings," by Prof. H. F. Rugan.

The British Association.

The Dundee meeting of the British Association will open on September 4, when the new President, Mr. **E. A. Schäfer,** F.R.S., will deliver his address on "The Nature, Origin and Maintenance of Life." A joint meeting of the Botanical and Zoological sections will also consider the question of the origin of life, and Professor Minchin (London) will give an account of his experiments in the fertilization of some lower forms of marine life. Professor A. Senier will preside over the chemical section and Professor A. Barr over the engineering section.

Market Prices.
August, 1912.

Tin opened at 204, dropped £2 during the first two days, recovered on the 7th and since rising, had reached 212 by the 20th, afterwards declining to 210, but better on 30th at 212-10s.

Copper fairly steady, opened at 78, and kept within £1 till the fourth week, when it had risen over £79, reaching £79-10, by the 28th. Closes at £78-15.

Hematite has been rising. Opening at 75/- it was 78/- on the 7th, 77/- on the 20th, closes at 77/9.

Cleveland has also been strong. Opened at 59/-, reached 60/- by 13th, 61/11 by the 22d, 62/9 by the 24th, and closes at 63/7.

Scotch Pig has followed the same upward movement. Opened 65/-, reached 66/- on 14th and closed at 67/7.

Lead was £19-15 on the 9th, and £20 on the 22d, making 20-5 by the 24th, and £20-10 to £21 by the 28th, closing at 21.2.6.

India rubber. Para fine. Has been fairly steady, with rise during last few days. Closes at 5/1.

	£ s. d.
Alum, lump, loose per ton........................	5.15.0
Antimony, black sulphide powder, per ton..........	20. 0.0
Borax, British refined Crystal, per ton.............	17. 0.0
Copper Ore, 10 to 25%, per unit..........14/1½d. to	14.7½
Copper Sulphate, per ton..........................	24. 0.0
Carbolic Acid, liquid 97/99%, per gal..............	1.6½
Creosote, ordinary good liquid per gal.............	.3⅛
Camphor, 1 oz. tablets............................	1.7¼
Caustic Soda Ash 48% ordinary, per ton...........	5.10.0
Hydrochloric Acid, per cwt........................	5.0
Naphtha, solvent, 90%, 160 deg. C., per gal........	1.2
Petroleum, Russian spot	.7¾
Quicksilver, per bottle............................	8. 5.0
Sal Ammoniac, lump, firsts delivered U. K. per ton...44. 0.0
Sulphate of Ammonia, f.o.b. Liverpool, per ton....	14.10.0
Sulphur recovered, per ton........................	5. 5.0
Shellac, Standard T. N. Orange spots, per cwt......	4. 4.0
Tin Ore, 70%, per ton.....................136 to 138. 0.0
Zinc, Vielle Montagne, per ton, Antwerp...........	29.15.0
Platinum, nominal, per oz.........................	9. 5.0

CORRESPONDENCE.

Function of Slag in Steel Refining.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The paper by Dr. Amberg in your September 12th issue on "The Function of Slag in Electric Steel Refining" is very interesting, since it touches on a subject which is of great importance and of which we still know relatively little.

The primary function of slag in an electric steel refining furnace is to remove the impurities—phosphorus, sulphur, oxygen, and, where desirable, manganese and silicon. In present practice the actual function of the "first" slag is practically the same as for an ordinary basic open-hearth slag. This fact is pointed out by Dr. Amberg in his interesting paper and this function might be called the "oxidizing slag function" in contrast with the "reducing slag function."

In view of the primary difference between an electric furnace and a combustion-heated furnace, making possible the complete control of the conditions in the electric furnace, it should be possible to give the refining slag a greater variety of functions in the electric furnace than in a combustion-heated furnace, and the slag should not only take up the given impurity, or several impurities, but should do this quickly and economically and without any detrimental action on the metal. That is, it should not leave the metal in such a state as to require further refining to correct the bad effect of the first slag.

In the present operation of electric furnaces a second slag is essential to correct the bad effects—namely, the over-oxidation resulting from the first slag.

What the writer wishes to point out is that in order to fulfill the functions of an ideal slag the conditions in the furnace must be neither extremely oxidizing so as to require corrective effect later nor extremely reducing so as to reduce the impurities back into the metal.

The slag may, of course, be used first as a carrier of oxygen as well as a fluxing agent for the impurity, but when this additional oxidizing function is required of the slag, some other of its functions—namely, its reducing action—is necessarily limited thereby, and usually made inoperative. Furthermore, the oxidizing action of a slag should be slower than the oxidizing action of a gas blown into the metal. In the latter case where the gaseous oxidizing agent is used it is possible to more nearly reach the conditions of an ideal slag, in which the slag acts purely as a collecting and holding agent—that is, a fluxing agent for the impurities without injurious effect on the metal.

Dr. Amberg refers to the use of carbon and silicon as the reducing agents for reducing the slag and deoxidizing the metal. From the writer's experience it has been found that a gaseous agent can be used to reduce oxides of iron without reduction of calcium phosphate, and the tests referred to in Dr. Amberg's article of using solid carbon to deoxidize a phosphate slag without reducing the phosphorus back into the metal have, so far as the writer knows, proved a failure. The use of a gas, however, can accomplish this result. An ordinary producer gas with six or eight per cent of CO_2 will serve to reduce the iron oxides out of a phosphate slag without reducing the calcium phosphate therein, and at the same time will permit that slag to take up and hold sulphur from the metal.

Chicago, Ill.   ALBERT E. GREENE.

* * *

To the Editor of Metallurgical and Chemical Engineering:

SIR:—It is to be regretted that as Mr. Greene was not present at the meeting of the Congress he could not contribute the interesting remarks on his process of deoxidizing by gases to the general discussion. As I have not had any practical experience with it I could not deal with it in my paper.

The process of Ernest Humbert to which Mr. Greene seems to refer implicitly has apparently proven a failure in the instance which Mr. Greene knows of, but it is giving excellent results right along in many other cases when properly carried out.

I would by all means keep the CO_2 of an "ordinary producer gas" below 6 per cent. This would mean that the Greene process necessitates a special producer and an independent gas main, viz., increased cost of installation.

Harrison, N. J.   RICHARD AMBERG.

Electric Steel Refining.

To the Editor of Metallurgical and Chemical Engineering:

Sir: Referring to your note regarding the Bethlehem Steel Company, on page 509, of your September issue, we should like to say, merely in order that there be no mistake, that Friedr. Krupp, of Essen, Germany, also has one other electric steel furnace besides the one you mention. They have a Frick induction furnace, which has been in operation for making high-quality special steels for the past three years. These special steels embrace not only the various tool steels, but also those used for high-quality forgings and castings. In addition it is used for melting ferro-manganese; its use in the molten state instead of its usual solid form, permits a saving of at least one-third of this alloy, when added to the Bessemer process steel. As for electric steel rails, these have been in operation in Germany for four years past in all sorts of weather and temperatures and have proven to be unbreakable in service.

K. G. FRANK,
Representing Siemens & Halske Company
and Siemens-Schuckert Company.

New York City.

The Power Required for Refrigeration.

By John J. Smith.

The use of artificial refrigeration for industrial purposes has increased to a great extent in the last two decades, and with the development of chemical industries of various kinds, is undoubtedly destined to increase much further.

Under these circumstances it is very necessary for those interested in such industries to know what the cost of such refrigeration will be, or, at least, what power it will require.

Engineers are very generally accustomed to thinking and calculating in units of horsepower, but relatively few of them are sufficiently familiar with refrigeration to be able to calculate, from the refrigeration required, the horsepower necessary. It is to supply, at least in part, this deficiency that the present article is written.

In the year 1906 the Iron and Steel Institute of Great Britain published a paper by J. E. Johnson, Jr., on which the present article is founded. This paper pointed out the connection as shown by thermodynamics between the quantity of refrigeration, the temperatures involved and the theoretical horsepower required, and then gave a method for determining the factor by which the latter must be multiplied to give the actual horsepower needed, based on the best data then available, notably the publications of Professor J. E. Denton, on that subject.

Within quite recent years, however, the York Manufacturing Company, under the guidance of its General Manager, Mr. Thomas Shipley, has conducted an enormous number of scientific tests, working under widely varied temperature conditions, and has determined the quantity of refrigeration, the pressures of ammonia, the temperatures corresponding, and the horsepower developed in the steam cylinders of the compressors for each set of conditions. These tests they have been liberal enough to publish in their catalog, so that the information is accessible to anyone.

When these data came into my hands I thought it would be interesting to check up the earlier work of Johnson and see whether his type of equation for the conversion factor between theoretical and actual results was correct. The results seem to me of sufficient importance to publish.

For the benefit of those who have not seen the earlier article, above mentioned, it may be well to explain briefly the formula which connects quantity of refrigeration and power requirements, but first it is necessary to point out a curious condition which has grown up in the refrigerating industries.

The earliest refrigerating machines were put in to take the place of refrigeration with natural ice. As a result they were all rated in tons of "ice melting effect," that is, a "ten-ton" machine would do the same amount of refrigeration as the melting of 10 tons of ice directly applied.

The heat absorbed by ice in melting is a perfectly well known and definite quantity, 142 B.t.u. per pound, or 284,000 B.t.u. per ton, and since refrigeration furnished by ice must be applied at the unvarying temperature of melting ice, 32 deg. F., no other factor than quantity of heat to be absorbed need to be considered. It was like pumping water from one lake with a fixed level to another at a higher level. Quantity was the only variable factor.

But with artificial refrigeration all this is changed because the temperature of application of the refrigeration may be anything from 50 deg. F. above zero to 50 deg. F. below zero, and the power required for a given quantity of refrigeration varies greatly according to the temperatures between which it is performed.

To utilize our illustration again it is like pumping from different wells, the water level of some at the surface, that of others, hundreds of feet below it. Height in this case becomes a factor whose variations are as important as those of quantity. But this change, so important not only in thermodynamics, but also in practice, has been frequently overlooked, and until recently it was customary to give the capacity of a refrigerating plant in "tons," with little or no attention to the temperatures involved. This, of course, is precisely the same as giving only the quantity of water pumped in our second illustration without stating the difference of level through which it was raised.

It is hoped that this article will not only serve as a convenient means of enabling the power required for a stated amount of refrigeration to be determined for the given conditions, but that it will make clear to all the direct and vital connection between the power required and the temperatures involved, and will show the direct application of thermodynamics to this subject.

All students of thermodynamics know that the formula $\dfrac{T_1 - T_2}{T_1}$ is reversible and applies in theory just as much to refrigeration as it does to development of power from heat. T_1 and T_2, in this case, are the temperatures at which the heat is received and discharged from the working substance, exactly as it is in the same formula for a heat engine, but in this case the heat is received at the lower temperature and discharged at
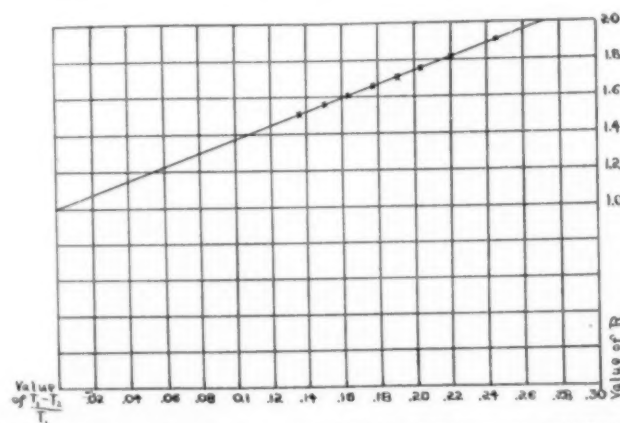

FIG. 1.—R IS RATIO OF 1 HP OF STEAM CYLINDER OF AMMONIA COMPRESSOR TO THEORETICAL OR DYNAMIC HP FOR DIFFERENT VALUES OF $\dfrac{T_1 - T_2}{T_1}$ PLOTTED FROM SHIPLEY'S TESTS.

the higher one, therefore, T_1, in this case, is the lower, and T_2 the higher temperature, thus being the opposite to the case of a heat engine.

Any quantity of heat "Q" measured in British thermal units per minute is converted into foot-pounds by multiplying by the mechanical equivalent of heat, 778, and this, in turn, is converted into horsepower by dividing by 33,000. Of this quantity Q the percentage transferred between temperatures $T_1 - T_2$ is given by the formula of thermodynamics $\dfrac{Q(T_1 - T_2)}{T_1}$ and this, in terms of horsepower, becomes H.P. $= \dfrac{778}{33,000} \; Q \; \dfrac{T_1 - T_2}{T_1}$

$$= .0235 \; Q \; \frac{T_1 - T_2}{T_1} \qquad (1)$$

The capacity of refrigerating machines being given in tons per twenty-four hours, this is reduced to thermal units per minute (Q) by multiplying by 284,000, and dividing by 1440, the number of minutes in a day. Thus, 197.+ B.t.u. per minute equals 1 ton of refrigeration per twenty-four hours. Therefore, if we have the temperatures between which a given refrigerating machine works and the number of tons of refrigeration that it performs, we can readily figure the theoretical horsepower required and can then compare this with the actual. The basis on which to make such a comparison is obviously on variations of $\dfrac{T_1 - T_2}{T_1}$ Accordingly, this quantity has been calculated for a number of the tests whose results are given by

Mr. Shipley, and also the ratio of the actual to the theoretical horsepower required for the same tests. These have been plotted in the diagram, Fig. 1.

Abscissæ are values of $\frac{T_1 - T_2}{T_1}$, ordinates are ratios of actual to theoretical horsepower. It will be observed that a line can be drawn through the points touching, or almost touching. most of them and passing through the point

$$R = 1, \quad \frac{T_1 - T_2}{T_1} = 0.$$

exactly as in the article by Johnson, though these data are more complete than were available to him. The equation of this line is $\quad R = 1 + 3.7 \dfrac{T_1 - T_2}{T_1}$ (2)

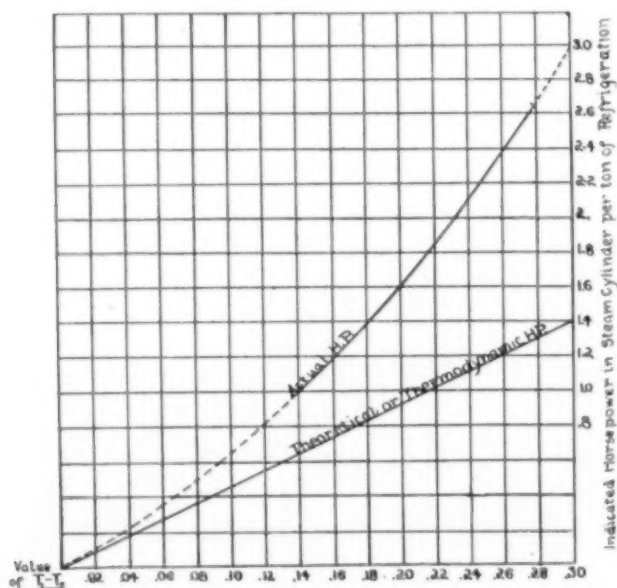in other words, the excess of the power actually required over

FIG. 2.—SMITH REFRIGERATOR, SHOWING THEORETICAL AND ACTUAL HORSE-POWER REQUIRED PER TON OF REFRIGERATION FOR

DIFFERENT VALUES OF $\dfrac{T_1 - T_2}{T_1}$

that theoretically necessary increases in exact proportion to the increase in $\dfrac{T_1 - T_2}{T_1}$, or, roughly speaking, it increases in proportion to the temperature difference, which is precisely what we should expect from our knowledge of the conditions.

Comparing this with the formula derived by Johnson from Denton's results, we find that he had $R = 1 + 4.15 \dfrac{T_1 - T_2}{T_1}$

The difference between 4.15 and 3.7 represents the increase in the efficiency of refrigerating machines since Denton's tests were made.

It is proper to point out also that the results given by the York Manufacturing Company are not the exact power requirements under test conditions in each case, but are about 10 per cent greater so as to represent simply reasonably good operating conditions.

It is believed that this formula will be a tool of real usefulness in the hands of those who will take the trouble to use it. If an engineer has the problem of cooling a given quantity of gas or liquid, or solid, through a given temperature in a given time, he can generally find a reasonably accurate value for the specific heat without much difficulty. This will enable him to figure the number of thermal units per minute to be removed. The temperature to which the given substance must be cooled will, of course, be known as one of the conditions of the problem. The temperature of the cooling water

available either from a natural source or from artificial air cooling, controls the temperature of the ammonia condensers T_2 in the formula. The latter temperature will always be about 10 to 15 deg. higher than the former. Similarly the ammonia in the cooling coils must be carried to a temperature about 10 deg. lower than that of the substance to be cooled, if direct cooling is used, or about 20 deg. lower if the ammonia is used to refrigerate brine, and this, in turn, is used to cool the substance under consideration. These, of course, are conditions almost as various and as numerous as the problems to be solved.

But if the temperature of the ammonia condenser and of the ammonia in the cooling coils be known or determined, and the quantity of heat to be removed be known, then the power required for the purpose in good standard practice can be calculated from the formula given above with all needful accuracy for industrial purposes.

For illustration of the above principles; certain reactions will only take place below certain temperatures, or if the temperature gets too high the consequence may be disastrous, as in the case of the manufacture of nitroglycerine. How much power is required to keep the temperature below the critical point?

In such a case the amount of heat evolved per unit weight of material is accurately known or can be accurately determined, and, of course, the weight of material is given. The temperature below which the reaction must be carried on has been found by experience.

Let us assume, in the case of nitroglycerine, that the latter is 50 deg. F., and let us assume that the quantity of nitroglycerine to be manufactured is such that the heat to be removed below this temperature is 10,000 thermal units per minute. Let us assume that it is necessary to use brine in the circulating pipes because the acid used in nitrating would attack iron or steel pipes, which are virtually the only ones that can be used for carrying ammonia. In order to force the heat through the walls of the pipes, let us assume that we must have a 12-deg. interval of temperature for each set of pipes; that is, the brine must be 12 deg. lower than the nitrating tank, and the ammonia 12 deg. lower than the brine. The temperature of the ammonia then becomes 26 deg. F. Let us assume that cooling water is available at the warmest period of the year at 72 deg., and that the corresponding temperature in the ammonia condensers is 85 deg.

It may be well to state here that I am not familiar with the details of the manufacture of nitroglycerine, and the above assumptions are not supposed to be based on actual conditions; they are merely used for purposes of illustration.

For this case T_1 becomes $26 + 460 = 486$, and T_2 becomes $85 + 460 = 545$. $\dfrac{T_1 - T_2}{T_1}$ becomes .121, for which the value of R from the formula or chart, is nearly 1.45.

The theoretical horsepower required, based on formula 1, in which Q becomes 10,000 and $\dfrac{T_1 - T_2}{T_1}$ has the value stated, is 28.6. Multiply this by 1.45, we obtain as the actual horsepower required in the engine cylinders, 41.4. This reduces the problem to terms with which engineers are perfectly familiar.

When this article had been completed to this point, it was realized that a still further simplification of this matter was possible, because both the theoretical horsepower required and the ratio R, by which it must be multiplied to give the actual horsepower, depend solely on $\dfrac{T_1 - T_2}{T_1}$. If, for convenience, we call this latter quanty Z, we have from (1) denoting by hp, the theoretical horsepower,

$$hp = .0235\ Q\ Z,$$

and from (2)

$$R = 1 + 3.7\ Z.$$

Denoting by H. P. the actual horsepower we have

$$H.\ P. = R \times hp = .0235\ Q\ Z\ (1 + 3.7Z).$$

This obviously gives: $\ H.\ P. = .0235\ Q\ (Z + 3.7\ Z^2).$

It is obvious from this formula that when the quantity of refrigeration Q and the temperatures are known, the horsepower required can be determined very readily by calculation.

But it can be still more easily obtained from a chart, plotted according to this formula, which is given herewith. The straight line marked "theoretical, or thermodynamic horsepower," gives the horsepower per ton of refrigeration per twenty-four hours required according to thermodynamics. The parabola tangent to this line at the origin and rising rapidly above it, gives the actual horsepower required per ton of refrigeration in any given case, according to the results of Mr. Shipley's tests and to the formula herewith.

Taking the value of $\dfrac{T_1 - T_2}{T_1}$ equal .121, in the case given above, we find the horsepower per ton from the chart to be .82. The quantity of refrigeration required in tons is the B.t.u. per minute (10,000) divided by 197, or 51. This multiplied by 0.82 gives us 41.4, exactly as before, without the use of any formula whatever.

The portion of the actual horsepower curve, represented by the solid line on the diagram, is that based on actual tests. The dotted portions at the ends are plotted from the formula, assuming it to be correct for temperature ranges higher and lower than those given.

From what has been said above, it will be seen that this is probably more than the reduction of empirical data to a graphic form, and that not only formula (1) but probably also formula (2) represents with close approximation a natural law, since it implies that the losses in a refrigerating process are closely proportional to the temperature interval involved, which is in line with our general knowledge of such matters.

It seems not amiss to point out out that this actual horsepower curve emphasizes a law which refrigerating engineers understand thoroughly, but which has not always been realized by others, and that is that the power required for performing a given amount of refrigeration increases enormously with the temperature-interval involved, and for that reason all refrigeration should be done at as high a temperature, be as direct, and involve as few transfers of heat from one medium to another, as the conditions of the given case will permit.

Adobe as a Building Material.—It is possible that in some places in the southwestern part of the United States, and in Mexico, where suitable building material is scarce, adobe may be used for the construction of small mine buildings. The mixture can be prepared by collecting the surface layer of prairie soil where the grass is as thick as possible, adding a suitable amount of water, and then mixing by causing horses to tramp through the mass. Some hand-mixing with fork or shovel may be necessary to insure breaking up lumps. When the adobe has the proper consistency it can be thrown into the wall between upright timbers spaced according to the dimensions of the building. After a layer 12 in. thick has been deposited, and has become hard but not dry, it is trimmed inside and out by laying a board on top of it as a guide, and cutting the adobe with a heavy knife, such as a hay-knife. The width of the board will be the thickness of the wall. Care should be taken to have the successive layers of adobe well cemented to each other to avoid flaws in the wall. Frames for doors and windows can be placed as the building progresses. Plates can be anchored on top of the walls and the roof frame attached to them. Adobe buildings are not likely to settle after they are dry, and they are comfortable.

Platinum was produced by only two of the United States in 1911, California and Oregon. The former produced 511 ounces and the latter 117 ounces. The value of the product was $18,138. Importations of platinum during the same period amounted to 27,500 ounces, nearly four times the domestic production. The world's production of this metal in 1911 was 314,323 troy ounces.

Alumina, Hydrochloric Acid, Caustic Alkalis and a White Hydraulic Cement by a New Process from Salt, Clay and Lime.*

By ALFRED H. COWLES.

Since 1862, or earlier, the proposal has been in the literature to produce hydrochloric acid and potassic or sodium compounds from salt and clay by passing steam over the mixture at a high temperature. These proposals have all been condemned as impracticable by Thorpe's Dictionary, Lunge and other high authorities. (Lunge, vol. 2, p. 429, 1895 edition. Thorpe's Dictionary, vol. 3, p. 454.)

It was not until 1902 that a method was patented by Adolf Kayser to effectively and economically separate the silica from an alkali-silico-aluminate formed as one product in the above process. (U. S. patent to Adolf Kayser, dated September 5, 1901; issued September 9, 1902.)

From another cause, the attempts to render this process a success when carried out on a commercial scale, resulted in failure. Namely, if moist clay and salt alone be mixed together, moulded in the form of perforated briquets, and after drying, be heated to a good red heat, but not high enough to melt the final product; and vapor of water be passed through the mass, a chemical action begins on all surfaces of each heated briquet. The vapor of water gives oxygen to the sodium of the salt, and hydrogen to the chlorine, thus hydrochloric acid passes off as a vapor, while sodium monoxide combines with the silica and alumina of the clay, forming sodic-silico-aluminate. This action

FIG. 1.—PARTIAL INSIDE VIEW OF SEWAREN PLANT.

slowly passes inward from the surfaces of all heated briquets. The conversion of the material from the outer surfaces inwards is slow. Kayser, in the nineties, found that it took 72 hours for the change to penetrate through a briquet wall, from its two sides, that was scarcely more than ½ an inch in thickness, and this in a furnace capable of containing many tons of such material.[1]

The result of this slowness of action involved a too long heating of the briquets, necessitating an abnormally large volume of nitrogen and fuel gases passing through the furnace to effect the result. The quantity of these inert gases was so great, that the hydrochloric acid fumes evolved were not capable of efficient and practical condensation.

Adolf Kayser died in Lockport, N. Y., on November 26, 1910, of heart failure at the age of seventy-three. He came originally

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*An abstract of a paper presented before the Section on Inorganic Chemistry of the Eighth International Congress of Applied Chemistry on September 12, 1912.

[1] Mr. Cowles exhibited one of Kayser's briquets. It has a crystalline structure. Though shrunken somewhat, it has retained its moulded form.

from the Hartz mountains, a simple yet energetic man in his habits, one loved by all and highly admired by those who could understand his clear philosophical reasoning. He, like Leblanc, failed to realize his thirty years dream of success. Unlike Leblanc, he was happy in his latter years in following his passion for original chemical research. He discovered a soluble sodic-silico-aluminate of high detergent value, now finding its way into commercial channels as a substitute for soap.

While he was thus employed as a chemist with the Electric Smelting & Aluminum Company, of which the author is president, I suggested that we make up some salt and clay briquets adding charcoal or sawdust to the mixture to render them porous during heating, and test them out in a comparable manner with similar briquets of clay and salt alone, such as he had formerly used in his attempts to render the process mentioned commercial. He did not think this mixture would work, as he was positive that it would destroy the oxidizing atmosphere needed in the operation to prevent the formation of volatile aluminium chloride (Al_2Cl_6. I believe that an oxidizing atmosphere was only necessary around the briquets, and that it would gradually penetrate the mass of each briquet, and that a reducing atmosphere at interior portions of such briquet could not interfere with the desired reaction, as any aluminium chloride formed would of necessity change to oxide before leaving the briquet. Without mentioning this to Dr. Kayser I insisted upon the trial being made. It was made. The bricks containing charcoal were converted into sodic-silico-aluminate in one-ninth of the time that was required with like briquets containing no charcoal. Seventeen deg. Beaume acid was condensed from a small apparatus in the face of imperfect control.

This experiment opened up an entirely new vista of success to the process. It meant that the volume of fuel gases and nitrogen necessary to pass through a charged furnace would become so reduced as to permit of efficient commercial condensation of the hydrochloric acid evolved. It meant a great saving of fuel and far greater capacity of output with the same cost of plant. The doctor and myself became equally enthusiastic. Our company acquired his patent for opening or rendering soluble the alumina in the form of sodic-aluminate from the sodic-silico-aluminate briquets produced, and after two years or more of added experimental work in the laboratory a large plant has been erected at Sewaren, N. J.

The process as a whole is easily understood. The clay, carbonaceous matter, and salt are made into perforated briquets and after being dried in a drying kiln they are passed on flat cars through a long furnace of the tunnel type which brings the briquets to a temperature of about 1035 deg. C.

Knowing the composition of the clay, and the percentage of salt in the mixture, steam is blown into the discharge end of the furnace over a train of cars covered with firebrick, with their iron parts sealed off, while through a vestibule at the other end of the furnace loaded cars are gradually pushed forward by a hydraulic pusher whose speed is under control. The weight and composition of the material on the cars thus fed into the furnace is known, and the correct proportional

weight of exhaust steam is passed into the furnace from the opposite end. From the charging end the gases at a temperature too high to attack iron are sucked by a large blower and forced through a large hydrochloric acid condensing system. The temperature of the furnace is controlled by oil heat and measured by pyrometers. Most of the heat required is furnished by the carbonaceous material mixed with the clay and salt of the briquets. The reaction that takes place will vary according to the composition and mixture of the aluminous material and the salt, but typically, the final reaction is as follows:

$$Al_2O_3 \cdot 2SiO_2 + 4NaCl + 2H_2O + xH_2O + xC + xO = (Na_2O)_2(SiO_2)_2Al_2O_3 + 4HCl + xH_2O + xCO.$$
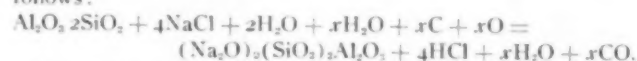
The surplus vapor of water and the hydrochloric acid gas are carried into the condensing apparatus and are condensed. The briquets maintain their original form as the cars bearing them are gradually pushed out of the discharge end of the furnace, having been cooled by the inflowing rush of measured
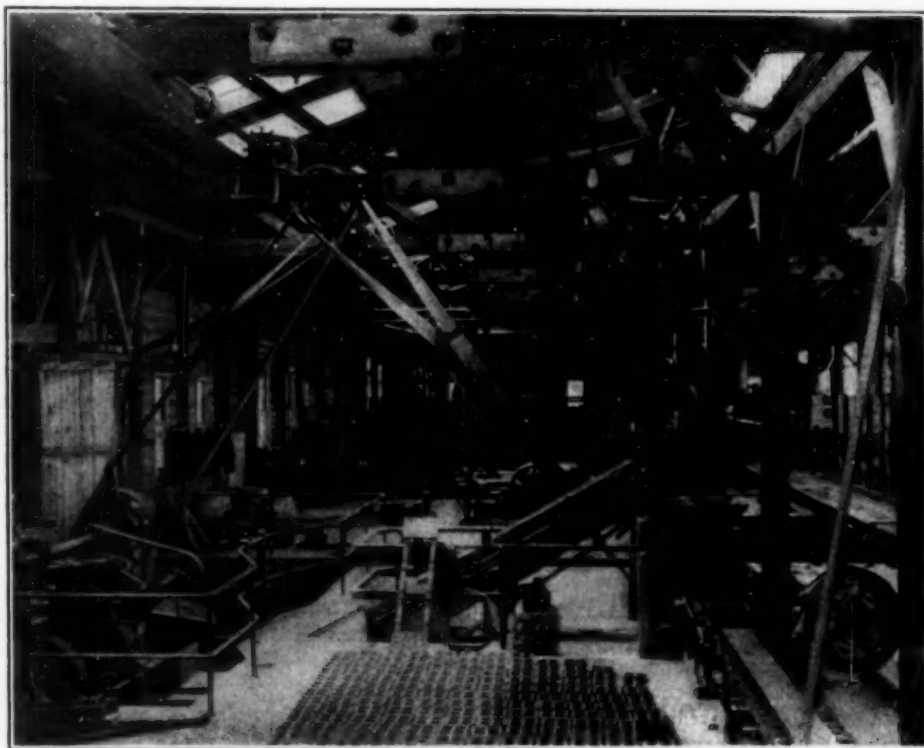
steam and a limited quantity of air while nearing the exit.

The sodic-silico-aluminate briquets formed have been made containing as high as 33 per cent of sodic oxide. The above reaction would give 35.8 per cent Na_2O. There is no appreciable loss of salt. While pure clays are preferred, all small amounts of iron pass off as iron chloride with the hydrochloric acid and are condensed with the first hydrochloric acid in the first condensers of a multiple series system. The acid produced is remarkably pure, free from arsenic, and but traces of sulphuric acid can come from iron pyrite if the same be in the clay. When briquets are made containing 32 per cent of sodic-oxide the output of 30 per cent hydrochloric acid should be about $1\frac{1}{2}$ tons of acid to 1 ton of salt and clay mixture charged into the furnace.

The next step in this process is to open the sodic-silico-aluminate, which is acid, to secure from it the sodic aluminate which it contains. This is accomplished by heating a mixture of ground acid salt with just sufficient lime to combine with the silica in it and form di-calcium silicate.

$$(Na_2O)_2 (SiO_2)_2 Al_2O_3 + 4CaO = 2((CaO)_2 SiO_2) + (Na_2O)_2 + Al_2O_3$$
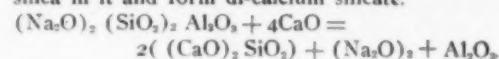
Limestone, calcite or marl in their chemical equivalent to lime may be used instead of anhydrous lime.

In the reaction just given, any amount over two molecular weights of calcic oxide to each molecular weight of silica, leads to formation of insoluble calcic aluminate, while less than two molecular weights is not sufficient and leaves undecomposed sodic-silico-aluminate. In each of these cases, marked loss in the recovery of alumina may occur. The mixture of lime and the acid salt is heated in a rotary cement furnace, or in a furnace of the tunnel type, as above described, just to its clinkering temperature. After its discharge from the furnace, it is broken into fragments and these are in an excellent physical condition for effective leaching. Very nearly all of the sodic aluminate may be leached from this clinker, while there is left behind a white mass of dicalcium silicate containing some sodic aluminate.

Having secured the sodic aluminate in solution, the method of procuring the alumina and caustic soda therefrom is that already in universal use.

The residue of dicalcium silicate when heated in a cement furnace with one molecular weight more of lime, gives a clinker that conforms in composition to Newbery's formula for the best and strongest hydraulic cement, but it has the advantage that all of the iron oxide has already been removed from the silica that goes into its composition, thus rendering it easy to secure a white cement by the use of limestone free from iron.

Such cement sells at about four times the market price of ordinary cement.

This process opens the way to secure the alumina cheaply,

est item of cost in producing this metal. By the general introduction of this process, we may hope to see the cost of production of the metal aluminium reduced four or five cents a pound.

This process also furnishes a key to unlock the stores of potash from the world's mountainous masses of potash felds-
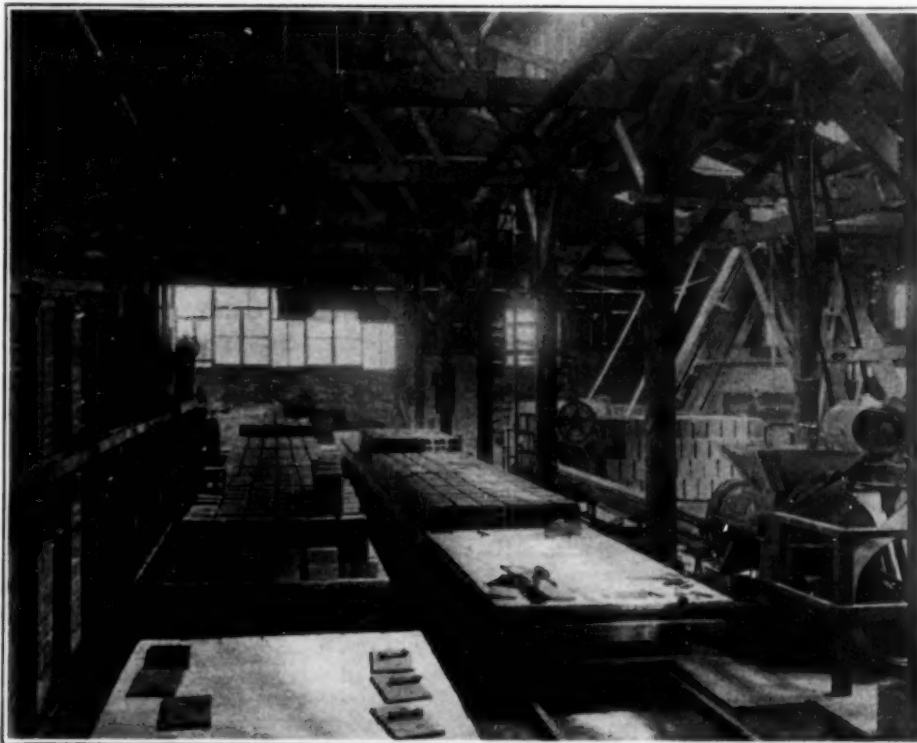

FIG. 3.—CARS WITH BRIQUETS READY TO GO INTO FURNACE.

par. As one way to do this effectively, the feldspar should be ground and mixed with either chloride of sodium or chloride of potassium and made with some clay and sawdust into the form of briquets, then passed through a furnace under conditions similar to those described above. The solid product should then be submitted to Dr. Kayser's lime opening process. The clinker on leaching gives potassic aluminate or a mixture of this with sodic aluminate from which the alumina may be separated, and the potash or potash and soda secured for use in fertilizers. The residue from leaching can then be converted into hydraulic cement.

A study of the reactions herein given in connection with the volatilizing temperatures of the various chlorides of the common metals will throw much light upon the geological theories of formation of intrusive mineral veins.

Before closing this paper, I wish to express my pleasure at being able to do homage to Mr. Kayser and make known the splendid work of an able en-

FIG. 4.—VESTIBULE WHERE CARS ARE ADMITTED TO LONG CHANNEL.

or as a by-product in conjunction with these other products, of great consumption in the industrial arts. There is required, 1.9 lb. of alumina, to produce 1 lb. of aluminium by the present method of manufacture. This costs 6½ cents and is the heavi-

thusiastic genial chemist, without whose aid the above described process as a whole would never have attained the completeness that it now has, and the great usefulness that future years will confirm.

Discussion.

In reply to a question whether any attempt has been made to catch and utilize the CO, Mr. Cowles replied that this had not yet been done, but may eventually be done as CO is utilized from blast furnaces. To remove small amounts of HCl from it, it should be passed through a proper scrubber.

In reply to a further question as to what use is made of white cement and whether the same has been made, Mr. Cowles said that white cement has high value for use in decorative work. They have, however, only made the dicalcium silicate. Adding one molecular weight of CaO to the same they know that they come within the formula discovered by Arthur New-bery for the strongest hydraulic cement, which formula permits up to several per cent of Al_2O_3 to be present in tri-calcium silicate, and if iron oxide be absent this produces a strong white cement.

Dr. A. S. Cushman asked whether the reactions are theories or have they been worked out on a large scale? Why does not the NaCl distil off? At what temperature does the final reaction take place?

Mr. Cowles in answering the first question said that the reactions have been worked out on a large experimental scale during three years work previous to their starting to erect the plant at Sewaren, N. J. He expected to have converted briquets from a channel furnace, 180 ft. long, at the meeting— the first from the new plant. But the day before an engine cylinder went to pieces that operated their generator, which furnished current to the motors driving the blowers connected with the furnace. This stopped the furnace with the bricks just approaching the hottest zone in the furnace. Acid was condensing in their condensing system at this time.

The NaCl is decomposed by superheated vapor of water before it has a chance to distil off. Some will vaporize in the hot zone of the furnace and be carried over the loaded cars. For about 90 ft. the hollow briquets act as a scrubber and carry the sublimed salt back toward the hottest zone of the furnace. Using too high a percentage of NaCl in the briquets might clog the furnace, for it would be capable of forming more Na_2O than the clay would be able to absorb. The limit above that required to place 33 per cent of Na_2O in the product will probably be higher with their large furnace than with the small furnaces that they formerly employed.

In answer to the third question, Mr. Cowles said that the reaction begins at a lower than the vaporizing temperature of salt. Two days before the meeting the hottest part of their large furnace was 1450 deg. F. and acid fumes so strong were passing through the condensing pipes that it was with difficulty that U-shaped pipes were connected between one set of upright acid ware pipes to permit the fumes to pass through other sets, and weak acid was drawn from the condensers. After the vaporizing temperature of NaCl about 1700 deg. F is passed, the reaction goes on very rapidly. The higher the Al_2O_3 in the clay the higher can the temperature be carried without melting the briquets.

Mr. L. P. Brown asked what purity of clay is required? Mr. Cowles replied that the purer the clay the better, as the presence of iron-oxide leads to a larger amount of iron chloride and loss of pure acid. If the iron chloride were used for leaching ores or otherwise, then its presence would not be so detrimental. The Moraine clays contain an undesirable amount of iron oxide and are most too low in alumina for obtaining the best results.

Mr. H. W. Charlton asked whether the process would work in an electric furnace with steam passed through a hole in the electrode?

Mr. Cowles replied that it would not work economically. Neither clay, silica or alumina will be fluxed by boiling salt. Vapor of HCl would come out of such a furnace and could be detected but it would be associated with sublimed salt and solid matter without complete transformation of the latter.

Resistance of Refractory Ware Against Abrupt Changes of Temperature.

By Paul Windszus.

In the manufacture of refractory ware, besides the usual criterions such as refractoriness, density and chemical constitution, account must also be taken of the capacity of resisting abrupt changes of temperature.

In order to establish how far the size of both grog and binding clay have any influence upon the resistance against abrupt changes of temperature, special tests have been made by means of fire bricks 25 cm. by 12 cm. by 6 cm.

The investigations, which were carried out at relatively low temperatures ($900° = 1000°$ C.), showed that by increasing the size of the grog particles and decreasing the percentage of binding clay in the mixture, the resisting capacity of fire bricks against abrupt temperature changes was essentially increased. No standard units for the measurement of a definite resistance against temperature changes have until now been proposed and no reference concerning the selection of those units can be found in literature bearing on the subject. It would, therefore, be advisable to establish standard rules from competent authority which would be accepted as a generally recognized standard for testing the capacity of resisting changes of temperature.

The suggestion for this inquiry was found in the fact that even high grade refractories often show a failure, either from being too porous or too dense; and that in the latter case they show more or less deep cracks. In both cases there is an uneven surface formed which offers less resistance to chemical agents and which shortens the life of the brick considerably.

Composition of the brick used during the tests.	Size of grains.	No. 1 Test. The air dry bricks were treated in a muffle at a temperature of 850-900° C. and cooled down in the open air.	No. 2 Test. These bricks which remained unaffected after the first t est were heated in a muffle at 900-950° C. for 20 minutes, then a jet of cold air was blown upon them for three minutes and were allowed to cool down in the open air.
Five parts of binding clay to five parts of grog.	0.75 mm to 6 mm	No cracks.	Small cracks.
	0.75 mm to 4 mm	" "	" "
	0.75 mm to 1 mm	Numerous big cracks.	" "
	0.00 mm to 0.75 mm	Parts of brick blown off.	" "
Three parts of binding clay to seven parts of grog.	0.75 mm to 6 mm	No cracks.	No cracks
	0.75 mm to 4 mm	" "	Small cracks.
	0.75 mm to 1 mm	" "	Numerous small cracks.
	0.00 mm to 0.75 mm	Small cracks.	

Translated from *Ton Industrie Zeitung* (*Berlin*), by E. C. Buck, St. Louis, Mo.

The temperatures used in carrying out the above described tests are far remote from the excessive temperature obtained in industrial and metallurgical furnaces. It was thought advisable to publish the above communication and results of the tests to our readers in the hope that the question entered upon by the author might be taken up for further elucidation.

Mining and Milling Machinery in New Zealand.—According to U. S. Vice-Consul General Henry D. Baker, the United States supplies New Zealand with only about one-fourth of its imports of mining machinery, and he considers that there is an opportunity for extending this class of business there. Machinery from British countries is admitted duty-free, but there is a tariff of 10 per cent on machinery from countries outside the British empire. A study of local conditions is urged as essential in seeking this trade, and local representatives are deemed advisable. The treatment of slime in the cyanide process offers an opportunity for the use of American machinery.

The Electric Steel Furnace in Foundry Practice.

By Paul Girod.

More and more the electric steel furnace is extending its influence in foundry practice as a useful aid to the open hearth and as a sharp competitor to the crucible or small converter.

This evolution is due to the possibility and ease of turning out a high-quality steel at a cost much lower than with the crucible process, and to the simultaneous advantage of producing a quality of steel superior to converter and open-hearth steel. Foundrymen are, therefore, becoming aware of the great possibilities of the electric furnace as a simple apparatus for making best-quality steel castings.

The electric steel furnace is especially well suited for castings since it offers the possibility of holding finished steel in the furnace as in a mixer and to tap it at intervals as desired into small ladles. Different taps can be made at different temperatures, lower temperatures being used for larger molds and higher ones for small castings. The steel can be heated to any desired temperature and castings can be made very thin-walled and of a very small size. Pieces from ½ lb. and upward with sections of 5 to 7 mm. thickness can be cast without any difficulty.

With the electric furnace one is sure to have for castings an absolutely dense steel without blowholes and pipes. Figs. 1, 2, and 3 relate to some small castings of extra soft steel.

The average results of analytical and mechanical tests of detached pieces of the castings were as follows:

C	Si	Mn	S	P	E	R	A	Charpy
0.08	0.05	0.37	0.022	0.017	24	56.5	32	24	Kilograms
0.10	0.32	0.43	0.022	0.017	25	40.7	28	23	Per square mm.
0.24	0.21	0.45	0.024	0.020	28	44.0	27	32	Formula for elongation.
0.31	0.22	0.48	0.018	0.018	28.7	48.1	25	20	
0.36	0.31	0.44	0.024	0.009	30.7	49.0	24	19.5	$L = \sqrt{66.67\ S}$

A cold-bending test is shown in Fig. 4.

Recently a very important improvement has been realized by introducing into electric steel practice for castings the use of special titanium alloys such as manufactured by the Société Anonyme Electrométallurgique Procédés Paul Girod of Ugine (Savoie), France, of approximately the following compositions:

I. Si : 56 per cent Ti : 36 per cent
II. Si : 21 per cent Ti : 28 per cent
III. Si : 10 per cent Ti : 10 per cent

Even very small additions of these alloys render the steel perfectly quiet, and insure the castings to become very dense and free from any blowholes. The number of lost heats can be very much diminished.

The microphotographs, Figs. 5 and 6 represent electric steel made with the aid of titanium alloys, and show the perfect density of the steel and the absence of any traces of oxides and blowholes. These microphotographs are magnified 220 diameters.

The construction of the Girod electric furnace is very simple. It is a hearth furnace formed by a rectangular or circular iron shell, and lined with refractory material. The furnace is fitted with a sliding door, through which the furnace is being charged and slag is being removed; opposite the door is the tap hole. The arched roof is made of silica bricks with metal framework.

The current is supplied by carbon electrodes hanging vertically into the furnace, their lower ends being near the slag level as in other furnaces.

In the bottom of the furnace and as far away as possible from the upper carbon electrodes, six or more steel electrodes pass through the refractory bottom and are also in direct connection with the furnace shell and the low-tension side of the power plant. The current is thus forced to pass through the charge. By this means the whole mass is uniformly heated, the bath not being simply licked on the surface by the arc. Moreover, the current passing through the bath produces an

electromagnetic field which gives the bath a rotary movement, accelerating the chemical reaction. The steel electrodes are water-cooled, and their lives are practically unlimited.

The steel frame is placed on rollers so that it can be tilted forward for the tapping or backward for the removing of the slag, which is done through the charging door. Since there is only one door an easy access to the furnace is assured and the entrance of too much air is prevented. The furnace is tilted by means of an electric motor or hydraulic plunger.

The furnace can be operated by any system of current, continuous or single-phase or three-phase current being suitable. The usual tension is 65 volts for small furnaces, and 70 volts for big ones. The frequency may be up to 50 periods, but on
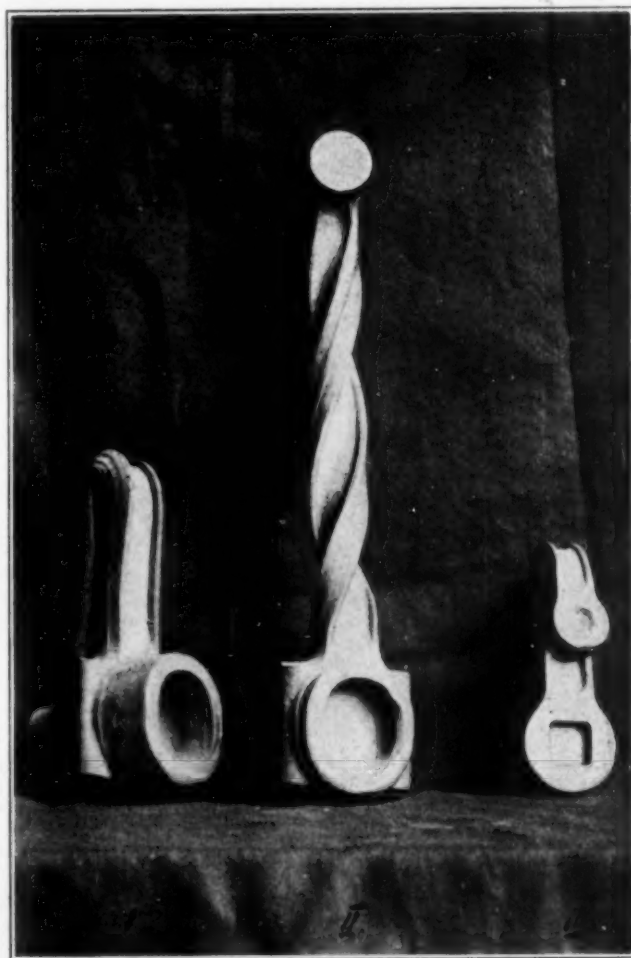
FIGS. 1 TO 3.—SMALL CASTINGS OF EXTRA SOFT ELECTRIC STEEL.

account of the effects of self-induction, it is much better to have a lower frequency, the electric efficiency being higher. On account of the low voltage of the furnace the workmen are not subject to any danger.

The use of single-phase alternating current is especially suitable for furnaces of small capacity with only one carbon electrode. Its use, however, entails in a three-phase distribution system in general the necessity of installing rotary converters or motor-generators with about 14 per cent transformation losses.

With a three-phase furnace merely the installation of step-down transformers is required with energy losses of only 3 or 4 per cent. For this reason, we recommend for operation by three-phase currents the use of furnaces with more than one electrode. Our special patented method of connections allows to feed our furnaces with three-phase currents with equal load on all three phases. With this kind of connections the current

passes through the whole bath, so that the great advantage of the Girod furnace, viz., the uniform heating of the whole bath is equally well attained in three-phase furnaces as in single-phase ones.

Figs. 7 and 8 show the general arrangement of a single-phase furnace and of a three-phase furnace respectively.

STEEL REFINING IN THE GIROD ELECTRIC FURNACE.

The Girod furnace is suitable for the melting and refining of ordinary scrap or steel waste of every composition. The
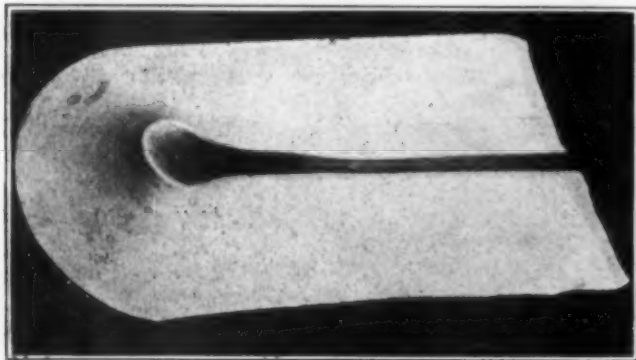

FIG. 4.—COLD BENDING TEST.

addition of pig iron to the charge is not necessary, and its use in small proportions offers advantages only in the case of highly oxidized raw material.

Metallurgical Reactions.—The operations can be divided into two periods:

1.—Oxidation period.

2.—Deoxidation period.

First Period.—An oxidizing slag is formed by adding to the charge some lime and iron ore. The proportion of these additions varies according to the chemical purity of the charge, and is determined in such a way that when the melting is completed the bath contains less than 0.1 per cent C, Mn and Si. In this period of refining the oxidation takes place vigorously and the original low temperature of the bath favors the elimination of the phosphorus. Practically all phosphorus is removed before slagging.

A forging test will easily show whether the metal is in the extra-soft state. In this case as soon as the temperature of the bath has become sufficiently high the slag containing phosphorus and iron oxides is tapped off through the charging door by tilting the furnace slightly backward. The first slag is immediately replaced by some lime which cleans the bath from the last traces of phosphoric slag. One slag is generally sufficient to remove all phosphorus down to traces.

Second Period.—The oxidizing and cleaning slag being removed, a first deoxidation of the bath is effected by adding deoxidizing agents, such as ferro-silicon, ferro-manganese, etc., these alloys being added in such proportion as not to remain in the bath. At that time recarburization takes place if hard steel is to be made.

Then the bath is rapidly covered with a slag consisting of about five-seventh of lime, one-seventh of silica sand, and one-seventh of fluorspar, together with a slight addition of carbon in the form of petroleum coke.

In this second period of deoxidation care must be taken that the furnace be properly closed, the temperature being held sufficiently high so that the iron oxide in the slag is reduced. The destruction of the ferrous oxide in the slag is indispensable for the deoxidation and desulphurization of the bath. It is gradually reduced by deoxidizers, such as ferro-silicon, and the addition of some petroleum coke to assure complete deoxidation, and a slag will be obtained which, in the air, disintegrates into a white powder, the ferrous oxide being reduced down to traces.

In order to push on the complete deoxidation of the bath it is useful to add small quantities of silico-manganese, or ferro-silico-manganese-aluminium, or even silicon-aluminium, these alloys acting very energetically upon the oxides in the bath, forming a very fluid slag which easily rises to the surface.

During the same period, when the slag is completely deoxidized and is very basic, the desulphurization which was incomplete during the first stage of the operation proceeds further, and is rapidly completed at the time of tapping. Desulphurization starts vigorously when the highly basic lime slag becomes white and readily fluid, *i.e.*, after the removal of the metallic oxides. Carburizing materials are eventually added for finishing the metal and the final addition of ferro-silicon, ferro-manganese, etc., is made in order to arrive at the required composition of the steel.

The average analysis of the final white slag is as follows: CaO, 74.85 per cent; SiO$_2$, 13.20 per cent; FeO, 0.13 per cent; MnO, traces; S, 1.2 per cent; Fe$_2$O$_3$, none; Al$_2$O$_3$, 1.75 per cent; MgO, 4.22 per cent; P$_2$O$_5$, 0.09 per cent.

COST OF REFINING.—COLD CHARGES.

Power Consumption.—The power consumption recorded at the terminals of the furnace and including smelting, refining, and finishing a charge of cold scrap is about

850 kw-hours per ton of metal tapped for a 3-ton furnace.

750 kw-hours per ton of metal tapped for a 10-ton furnace.

Of course, these figures vary with the composition of the charge. According to the quality of the steel and its purity, the figures may be higher or lower than those quoted.

FIGS. 5 AND 6.—MICROPHOTOGRAPHS OF STRUCTURE OF ELECTRIC STEEL WITH TITANIUM TREATMENT.

Consumption of Electrodes. — The consumption of electrodes per ton of steel made is about

8 to 9 kilograms for a 3-ton furnace,

8 to 10 kilograms for a 10-ton furnace.

These figures indicate the consumption of electrodes of good quality, as manufactured in our own electrode factory at Ugine, and may be considerably reduced by the latest improvements introduced in the manufacture of the electrodes which permit them to be completely used up.
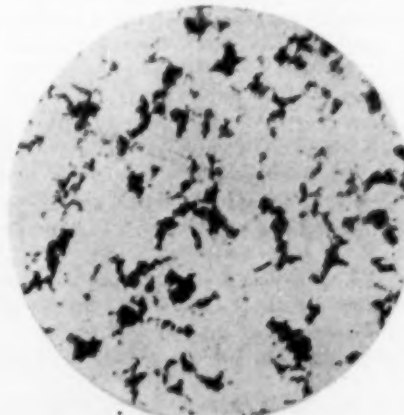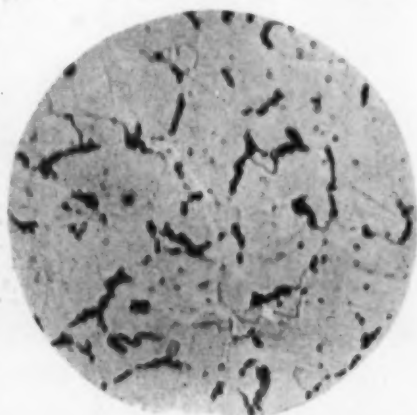
Wages.—The furnace needs:

1 smelter, 1 second hand, 1 boy for the 3-ton furnace.

1 smelter, 2 second hands, 1 boy for the 10-ton furnace.

Maintenance and Repairs.—The furnace can be lined with magnesite bricks or better tamped with tarred magnesite or

dolomite. Dolomite-tamped bottoms are giving the best results. The work is done with heated hand tampers or with pneumatic rammers. The life of the lining is about 90 to 100 heats for a furnace of 10 tons and about 120 heats for the 3-ton furnace. At the end of that time the side walls and the hearth will need repairing. All the burned or oxidized parts of the walls are scraped. The bottom is broken down about 10 to 15 cm. and is retamped with new layer of dolomite, taking care to leave intact the passage of the tops of the bottom electrodes. This repair of the upper part of the bottom is the only one required, the damage of the walls at the slag line being repaired between charges with dolomite.

The roof of the furnace is made of silica brick and stands on an average fifty heats for the 10-ton furnace and seventy heats for the 3-ton furnace.

Additions.—Lime, ore, silica, sand, fluorspar, petroleum coke, to make the slag, and ferro alloys for deoxidation and finishing.

The total of this expense varies considerably with the purity of the raw material used and the quality of steel.

FIG. 7.—SINGLE-PHASE GIROD FURNACE.

Heating Losses.—The output is about 90 to 96 per cent of the charge; it depends much on the degree of the oxidation of the scrap.

COST OF REFINING—LIQUID CHARGES.

With liquid charges, on the other hand, where the working period for a heat is only 2 to 2½ hours, the cost of electric steel refining is considerably reduced and allows a reduction of more than 25 per cent on the cost price according to the local conditions.

On the lines of these conditions and indications we arrive at the following *estimate for the cost price per ton of steel tapped with our system:*

COLD CHARGES.

Raw Material:

	3-ton.	10-ton furnace.
Scrap, 1100 kgs. at frs. 75 per 1000 kgs.	82.50	82.50
Slag	2.30	2.30
Deoxidizing additions and recarburization	3.50—88.30	3.50—88.30

Producing Cost:

Electric power, 850 and 750 kw-hours at 2 centimes	17.00	15.00
Electrodes at frs. 320.00 per ton	3.00	3.50
Wages	3.00	1.50
Maintaining and repairs	12.00—35.00	8.00—28.00
Total cost per ton, francs	123.30	116.30
or	$24.66	$23.26

MOLTEN CHARGES.

Raw Material:

	3-ton.	10-ton furnace.
Liquid steel 4 per cent loss in heating, 1040 kgs. at frs. 80 per ton	83.20	83.20
Slags	2.00	2.00
Deoxidizing additions	3.50—88.70	3.50—88.70

Cost of Producing:

Electric power, 275 and 200 kw-hours at 2 centimes	5.50	4.00
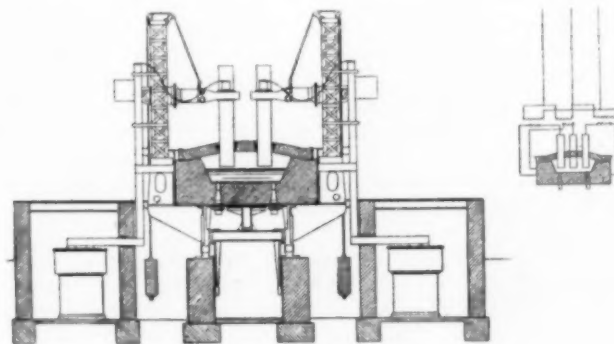
FIG. 8.—THREE-PHASE GIROD FURNACE.

Electrodes 3 to 4 kgs. at frs. 320 per ton	1.25	1.25
Wages, 8 heats in 24 hours	1.00	1.00
Maintenance and repairs of the furnace	4.00—11.75	2.50—8.75
Total cost per ton of steel, francs	100.45	97.45
or	$20.09	$19.49
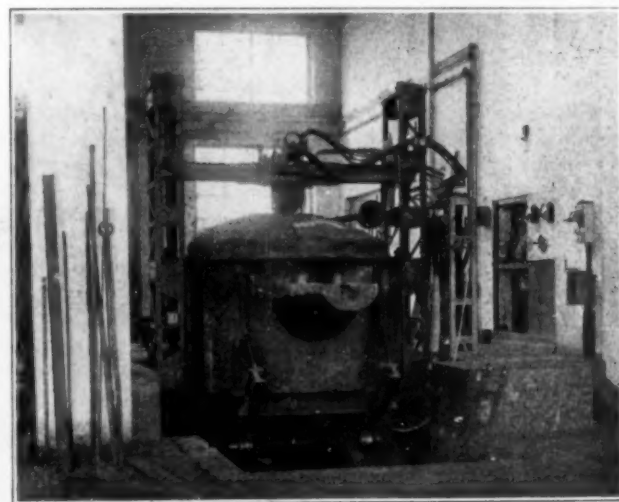
FIG. 9.—TWO-TON GIROD FURNACE FOR STEEL AND MALLEABLE CASTINGS AT STOTTZ WORKS, STUTTGART, GERMANY.

The estimate does not include the expenses for ingot molds, superintendence, laboratory, amortization, and general charges which vary so considerably as not to allow a fair average estimate.

Ugine, Savoie, France.

Alumite, which promises to become an important source of potash in the United States, is found in the San Cristobal quadrangle in Colorado. Analyses of the mineral indicate that it exists in probable commercial quantities.

Coarse Crystallization Produced by Annealing Low-Carbon Steel.

BY R. H. SHERRY.

The production of large crystalline grains during the annealing of low-carbon steel is a phenomenon often observed in plate sheets and other similar material which has been annealed at temperatures between 650 and 690 deg. C. One of the earliest references to the subject is by Stead in his "The Crystalline Structure of Iron and Steel," published in the *Journal of the Iron and Steel Institute* for 1898, Vol. 1. In this article he makes the following mention of this phenomenon:

"In practically carbonless pure iron and steels of fine grain produced by forging or certain heat treatment, the grains increase in size slowly at 500 deg. C. and more rapidly at between 600 and 750 deg. C., and it is possible by heating at about 700 deg. C. for a few hours to develop granular masses of exceeding coarseness. . . . . As soon as the temperature rises to above 900 deg. C. the granules again become small, and heating to 1200 does not apparently produce any difference in their dimensions."

In Vol. 2 of the same journal for 1898, Stead, in his paper "Brittleness Produced in Soft Steel by Annealing," makes mention of the same phenomenon in soft steel sheets containing 0.05 to 0.12 per cent carbon annealed for forty-eight hours. Laboratory attempts to produce this structure at will were not invariably successful.

As a further advance in the study of this phenomenon came the work of Charpy (*Comptes Rendus*, Vol. 151) showing that this coarsely granular structure could be produced in steel low in carbon and phosphorus by performing cold mechanical work on the steel and subsequently annealing at 700 deg. C. The specimens were worked by means of a screw plate and examination of them after annealing showed that the grains had increased to about ten times their original size. He also found that there was no limit to the size of the grains produced by annealing at the proper temperature, which varied with the nature of the steel, and that by annealing long enough the entire specimen could be converted into a single grain.

The object of this paper is to give a brief outline of the results of some experiments conducted with the idea of throwing some light on this phenomenon of the coarse crystallization of annealed low-carbon steel. In this work the temperature ranges noted by Stead were partially confirmed, and also the effect of cold mechanical work found by Charpy, but it was not possible to produce all the phenomena noted by them. This is especially true in regard to the increase in grain size due to long annealing. Nearly all the test pieces used in this investigation were annealed for periods of from one to forty-eight hours and no variation in grain size was found due to the increase in the time of annealing. It was found, however, that it was necessary to anneal for at least forty-five minutes to produce the grain growth. Nor was it possible to obtain the very large grain mentioned by them. The largest grain size obtained in this work was 0.80 mm. linear dimension, an increase to about forty times the original size. The photo-micrograph accompanying, taken with 100 diameters enlargement, shows the size of the grain in this specimen. Under the conditions by which this was produced, a matter which will be mentioned later, it was not possible to remove all the fine grain structure, and a number of grains which remained unacted upon will be seen in the photograph and serve by comparison to give some idea of the increase in grain size.

The first experiments were undertaken with steel which was known to crystallize when annealed at about 700 deg. C. The samples used were 0.3 in. in diameter. Of the previous history of this steel nothing was known except that it had received cold mechanical work. The analysis was:

Carbon.	Manganese.	Sulphur.	Phosphorus.	Silicon.
0.08	0.34	0.032	0.010	0.020

The grain size before annealing was 0.02 mm.; after annealing this increased to 0.52 mm. With this steel the temperature range within which crystallization would occur was found to lie between about 690 and 775 deg. C. Heating to 800 would remove any crystallization and would prevent its occurrence if the steel was subsequently annealed at 700 deg. C. On annealing below 690 there was no grain growth.

To study the effect of mechanical work a series of forging tests were made with this steel. It was found that on heating to various temperatures from 800 to about 1200 deg. C., to remove the tendency to crystallize, and then hammering the piece until the temperature had fallen to about 700 deg., no crystallization could be produced by subsequent annealing at 700 deg. If, however, the temperature was allowed to fall well below 700 before forging was stopped it was found that an increase in the grain size would occur if the sample was given the usual annealing at 700 deg. This grain growth, however, was confined to an area about ⅛ in. in from the circumference. In no case was the increase as large as that obtained with the steel in its original condition, but varied from 0.10 to 0.27 mm. On hammering cold a sample of the steel and subsequently annealing it was again found that the grain size did not increase as much as in the unhammered steel. Repetition of this experiment gave grain sizes of from 0.12 to 0.30 mm.

Further forging tests were made on samples of hot rolled steel of various sizes with practically the same result. Forging below about 700 deg. and subsequent annealing would cause the steel to crystallize around the circumference. Preliminary test showed that none of the hot rolled test pieces would crystallize on annealing. As found in the experiments with the first steel there was a considerable variation in the grain size of the forged and annealed test pieces. These results are purely qualitative and gave no key to the cause of this variation in grain size, and in order to obtain more quantitative results it was decided to try the effect of cold drawing and cold rolling where the amount of work could be determined in terms of the reduction of area.

In order to try the effect of cold drawing a series of hot rolled rods varying in size from ¼ in. to ½ in. in diameter were drawn with varying reductions of area and then annealed for one hour at 700 deg. C. The results of these experiments showed that in order to produce this increase in the grain size it was necessary that the amount of mechanical work be quite light. In general the lighter the work performed the larger the grain size. There was a limit to this, however, and no result was obtained with extremely light drafts. In general it was necessary to have at least 7 per cent reduction of area in order to obtain any increase in the grain size, though often with lighter drafts followed by annealing there would be a slight increase confined to the outer area, similar to that obtained in the forging tests.

Samples of all the drawn test pieces were annealed at various temperatures between 600 and 900 deg. C. The results of this experiment showed that if the reduction had been about 9 per cent or less crystallization would occur only when the steel was annealed between about 690 and 775 deg. C. On the other hand, if the amount of work corresponded to more than 9 per cent reduction, crystallization would occur between 650 and 875 deg. C.

To illustrate the effect of mechanical work four series of tests have been chosen from the experimental results, and are given in the appended table.

Another series of tests was made on cold rolled material, the results being practically the same as obtained with cold drawing. It was not possible, however, to obtain quite as large grains, and some variation was noted in the annealing range, as all the cold rolled specimens showed increase in grain size when annealed between 700 and 875 deg. C. No increase could be obtained by annealing at 650 deg., and heat-

ing to 900 removed crystallization and all tendency to crystallize. Two series are given in the table.

On examination of the results obtained by these experiments it will be seen that a number of variations from the results of Stead and Charpy were found, although the general features of the results of cold mechanical work are the same. The quantitative effect of cold mechanical work in the form of cold drawing and cold rolling have been measured and the connection with the temperature range outlined.

The composition of the steels used in these experiments was as varied as would be expected in low carbon steel of any purity, and the variations were as follows:

Carbon ....0.07 to 0.11 per cent.
Manganese ....0.30 to 0.70 per cent.
Sulphur ....0.02 to 0.04 per cent.
Phosphorus ....0.008 to 0.05 per cent.
Silicon ....0.000 to 0.075 per cent.

There were many features of this crystallization in low carbon steel which could not be covered by this investigation, but it is hoped that these results may serve to throw some light on a problem that has remained obscure for so long a time.
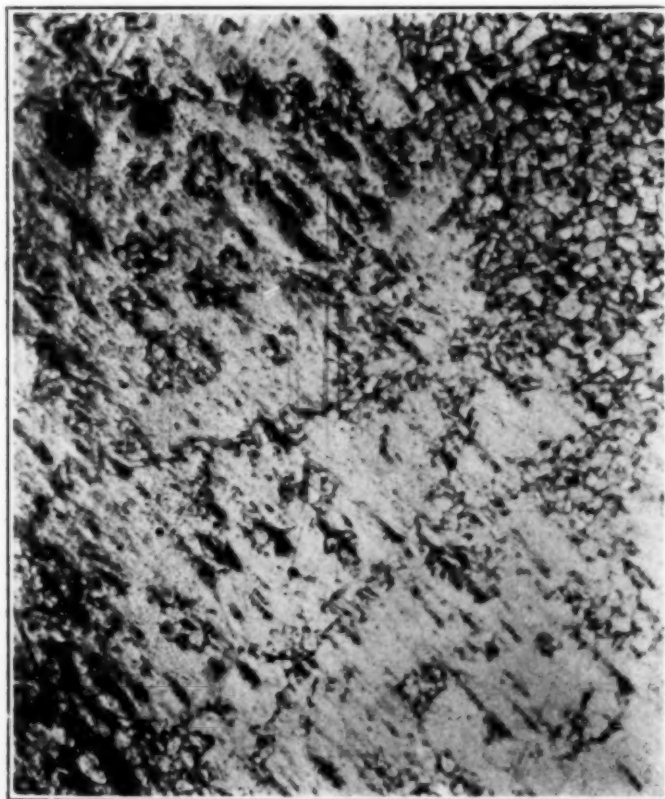

FIG. I.—SIZE OF GRAINS 100 DIAMETERS.

Table showing the results of varying amounts of cold mechanical work on the grain size of low carbon steel annealed in certain temperature ranges, and its effect on these temperatures.

COLD DRAWN.
Original diameter, 0.386 in.

Diam. after drawing, in.	Red. of area, per cent.	Grain size in mm. after one hour at 700 C.	800 and 850 C.
.372	7	.02	.02
.368	9	½, .02; ½, .30 to .52	.02
.3645	11	.27, a few .02	.27 to .02
.3595	13	.15	.15

Original diameter, 0.320 in.

.309	6.5	.02	.02
.307	8	A few .40 to .80 and .02	
		Rest .40 to .52	.018
.305	9	.28 to .52	.018
.303	10	Some .036, rest .40 to .50	.28 to .40
			Some .02
.302	11	Some .02, rest .036 to .15	.18
.300	12.5	.14	.14

Original diameter, .312 in.

.302	7	.52	.02
.296	10	.10	.10
.289	15	.06	.06
.280	20	.04	.04

Original diameter, .473 in.

.452	9	.27	.27
.445	12	.18	.18
.432	16	.07	.07
.419	22	.04	.04

COLD ROLLING.
Original hot rolled flat, 0.0625 in. in thickness.

Size after cold rolling, in.	Red. of area, per cent.	Grain size in mm. after one hour at 700 to 875 deg. C.
.059	5.6	.034, a few .10
.0575	8	.03, a few .14 to .27
.056	10	.14 to .28
.0545	13	.09 to .14
.053	15	.07 to .11

Original hot rolled flat, 0.64 in.

.061	4.5	.02 to .027
.0585	8.5	.02 to .27
.0575	10	.21, some .02
.056	12.5	.14
.054	15.5	.10
.0515	19.5	.055 to .082
.049	24.5	.027 to .036

Buffalo, N. Y.

Monel Metal's Resistance to Corrosive Action.—Recent experiments made at the laboratory of the Board of Water Supply, New York City, indicate that Monel metal possesses about the same resistance to corrosive action as the better known bronzes, while it had the additional advantage that it presented the least change in appearance as result of the corrosive action. Specimens of several bronzes, Monel metal and steel were weighed and embedded in rich earth which was kept wet for six months by periodical additions of very dilute solutions of corrosive salts. At the end of the test period all of the specimens were taken out, scrubbed, dried and weighed to ascertain the comparative loss from corrosion. The results were as follows:

	Per cent loss.
Phosphor bronze	0.09
Tobin bronze	0.11
Monel metal	0.12
Parsons manganese bronze	0.12
Muntz metal	0.33
Steel	1.04

Another test of the same kind under somewhat different conditions but the same period, gave about the same relative results. Because of its resistance to corrosion Monel metal is now being used in the manufacture of tanks for developing moving picture films.

The production of cyanamide by the American Cyanamide Company, of Niagara Falls, Ontario, is to be increased. The present plant is to be enlarged to more than twice its present capacity, at a cost of $500,000. The company has constantly increased its exports of cyanamide since the beginning of operation in 1910.

The Methods of the United States Steel Corporation for the Commercial Sampling and Analysis of Pig Iron.[1]

PREFACE.

In conformity with the design of the officials of the United States Steel Corporation for the standardization of the methods employed in the sampling and analysis of all materials encountered in their various lines of manufacture, the Chemists' Committee presents this compilation of standard methods for the sampling and subsequent analysis of molten pig iron.

In selecting the methods, the committee employed the same line of procedure as in former cases, viz., the careful consideration of all the methods employed in each laboratory of the corporation, evolving therefrom the several methods herein described, the immediate adoption of which is desired.

The services of Messrs. W. B. N. Hawk, Wm. Brady and C. H. Rich, a sub-committee appointed for the preparation of this pamphlet, are gratefully acknowledged.

INTRODUCTION.

A quite obvious cause of non-conformity of results of comparative analyses in the various laboratories of the corporation has been an apparent lack of uniformity of method in the sampling and analysis of molten iron.

This condition is particularly apparent with regard to the determination of the sulphur, numerous discrepancies in which would appear to indicate the absence of, and the necessity for, some uniform method of procedure. The estimation of the sulphur by the gravimetric method and the determination of the remaining constituent elements of the iron, are susceptible of a most satisfactory degree of accuracy. The successful issue of the volumetric determination of the sulphur, however, is largely dependent upon the grade of the iron and the conditions under which the test is taken. A disturbing factor in establishing a satisfactory standard of comparative value in such analyses is the loss of uncertain and varying proportions of the sulphur, due to the formation of volatile and difficultly oxidizable organic sulphides, upon solution of white or chilled iron in hydrochloric acid. Comparative tests by a number of chemists of the corporation having shown quite satisfactory agreement in sulphur values as obtained by the evolution method in sand cast and properly taken so-called chilled test samples, it has been decided to adopt for all purposes the more convenient chilled test, which, it is believed, will tend to conserve the desired comparative accuracy. True values may be obtained in the crushed shot sample by annealing the powder in a current of natural gas or under convenient non-oxidizing conditions, but the exigencies of the routine of the works laboratory, as demanded by the conditions necessarily involved in the manufacture of iron and steel, would in most cases prohibit this metallurgical refinement.

The methods of analysis herein described represent the best present-day practice and it is hoped that their adoption will go far toward reconciling those too often variant conditions, convenience of manipulation and accuracy of result.

The solutions as prescribed are indicated in their various proportions on a 1-litre basis. It will be found convenient, however, in most cases to employ in actual practice corresponding multiples of a greater volume.

Sampling.
Character of the Sample.

Two optional methods are prescribed for taking the test:

1. *Plate or Pat Test.*

With a suitable hand ladle a portion is dipped from the stream of molten iron and, with the spoon of the ladle resting upon a clean, dry iron plate, a pat of such size as may be desired is poured thereon.

2. *Chilled Mold Test.*

With the spoon of the hand ladle resting upon the dry, clean mold (later to be described) a portion is poured sufficient to fill the mold.

The use of the water shot sample is to be considered as in violation of the standard method. The sand cast sample may be used, but the uniform adoption of the pat or the chilled mold test is urged.

Number of Samples to be Taken.

As tending to a reasonable degree of accuracy, it is recommended that a portion be secured, timed as nearly as may be practicable for the middle of each ladle of iron representing the cast. Equal portions from each of these samples are conveniently combined for the subsequent analysis.

Preparation of the Sample.

The tests are either drilled or crushed as required. If crushed, only that portion passing an 80-mesh sieve is used for the subsequent analysis.

Description of the Mold.

Two convenient forms of mold are recommended, either of which may be used to advantage.

The mold (Fig. 1) furnishes a test which may be conveni-

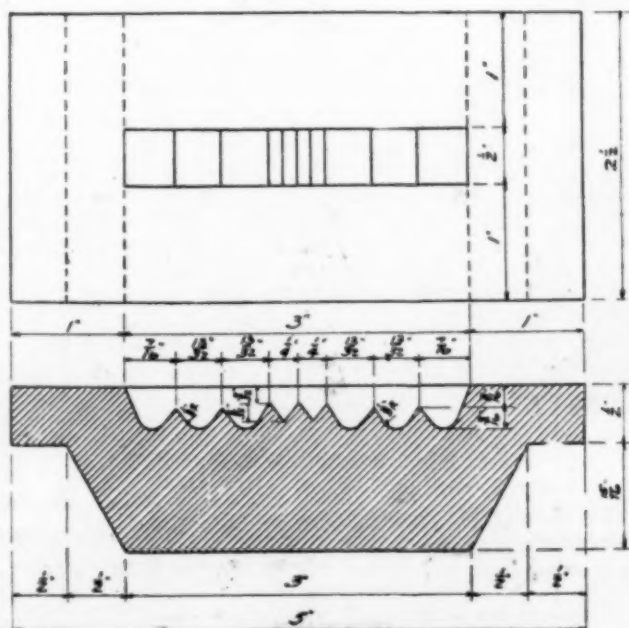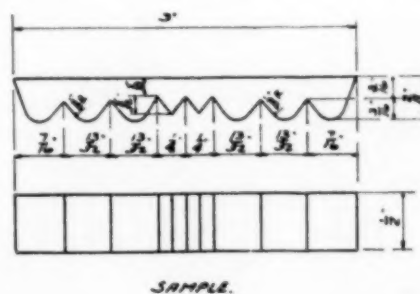SAMPLE.

FIG. 1.—IRON MOLD AND TEST.

ently broken and crushed. The number of the furnace may be cast in the mold with Roman numerals or any other suitable form of designation. The two ridges shown in the test in this figure indicate that the test is taken from furnace No. 2.

The mold (Fig. 2) furnishes a test which may be either drilled or crushed. If the sample is to be crushed, the notched

prolongation "T" is broken and crushed and the main body of the test reserved for the furnace man's fracture test. The furnace number may also be indicated in this form of mold in the same manner but not so conveniently, and only in the main body of the test.

The molds are made of cast iron and are illustrated in detail, with complete dimensions, in Figs. No. 1 and No. 2.

Methods of Analysis.
Determination of Silicon.
Drown's Method.

A single or double factor weight, .4693 or .9386 gram, of the sample is transferred to a glazed porcelain dish, 15 to 25 cc. of the silicon mixture added, gentle heat applied to the uncovered dish, and the solution is evaporated to pronounced fumes of sulphuric anhydride. The dish is removed from the heat and allowed to cool, the residue is moistened with 10 cc. of the dilute hydrochloric acid, 50 cc. of hot water added and the solution heated without boiling until the soluble salts dissolve. The residue of silica and graphite is collected on a 9 cm. ashless filter, washed alternately with hot dilute hydrochloric acid and water to the disappearance of the soluble iron, and finally with hot water until free from chlorides. The residue is ignited uncovered in a weighed crucible at a moderate temperature until the carbon is completely oxidized, and finally for a few minutes at a higher temperature to the dehydration of the silica. The residual weight in milligrams divided by 10 and corrected in conformity with the initial weight of the sample equals per cent silicon.

Ford's Hydrochloric Acid Method.

This well-known method is useful in point of rapidity, yielding satisfactory results in a shorter time than the more refined Drown's method. A single or double factor weight of the sample is transferred to a porcelain or platinum dish with cover glass, 20 or 30 cc. of concentrated hydrochloric acid added and the solution boiled rapidly to complete dryness; the residue is not baked. Ordinarily, decomposition is complete by the time the evaporation is effected. To the dry residue in the platinum dish there is added at once, or in the porcelain dish when sufficiently cooled, 20 cc. of the dilute hydrochloride acid. The solution is heated gently for a few minutes with the addition of 50 to 60 cc. of water, and the heating continued until the soluble salts dissolve. The residue is collected and finally weighed as in the previously described Drown's method. The use of suction in the filtrations and the ignition of the residue in an atmosphere of oxygen will be appreciated as factors affecting the rapidity of the analyses in both of the above methods.

SOLUTIONS REQUIRED.
Silicon Mixture.

Nitric acid	1.20 sp. gr.	750 cc.
Sulphuric acid	1.50 sp. gr.	250 cc.

Dilute Hydrochloric Acid.

Hydrochloric acid	1.20 sp. gr.	500 cc.
Water		500 cc.

Determination of Sulphur.
Volumetric Method.
Evolution Iodine Titration.

It is apparent that the evolution method is of comparative value only and is in no sense to be regarded as yielding the exact measure of the sulphur content of the iron, a fact of which due cognizance should be taken by all users of pig iron. Equally apparent, however, is the undoubted usefulness of the method in connection with the routine requirements of the iron laboratory and of the various metallurgical processes in conjunction with which it is used.

The apparatus as recommended for this determination consists of a flask of suitable form and size, preferably 16-ounce Florence ring neck with a two-hole rubber stopper equipped with an ordinary funnel or thistle tube and a short piece of glass tubing bent at right angles (exit tube) in the respective apertures. The exit or delivery tube is connected with the absorption vessel in such manner as to conduct the evolved gases to the bottom of the vessel. The absorption may be effected either in the beaker in which the titration is to be conducted, or in a test tube, preferably 10 in. by 1 in., suitably supported.

A minimum of 2.5 grams or a maximum of 5 grams of the sample is transferred to the flask, the stopper is inserted and the flask conveniently connected with the vessel containing the absorbent; 40 to 60 cc. of dilute hydrochloric acid are introduced through the funnel and heat applied, a brisk evolution of the gases being maintained until the powder is decomposed. The temperature is then raised until the steam formed has forced the last trace of the hydrogen sulphide gas into the absorbent. The heat is removed, the apparatus disconnected and the titration conducted.

Ten cc. of the starch solution are added to the titrating beaker and the contents of the absorption tube transferred

SAMPLE

SECTION A-A

SECTION B-B
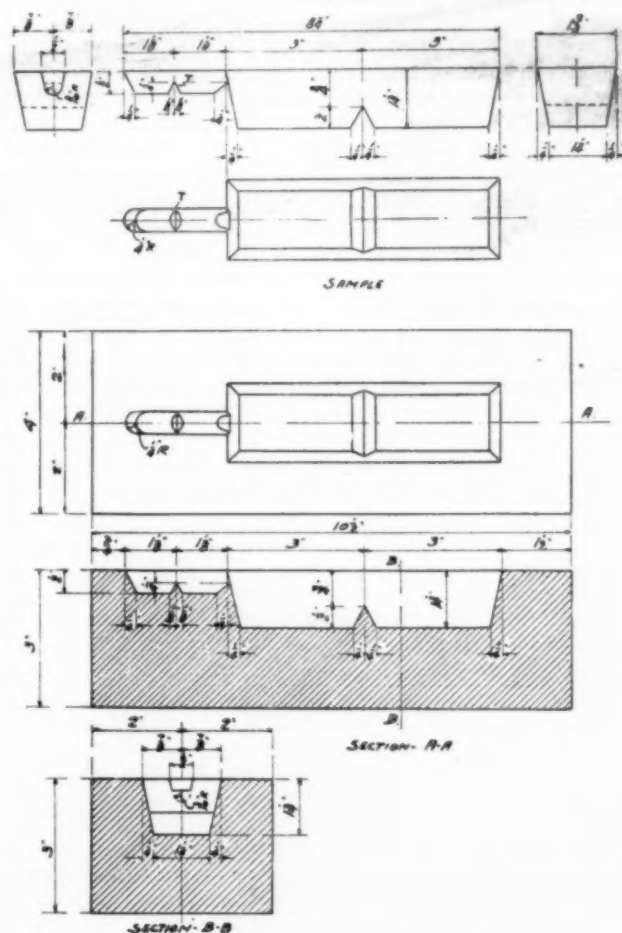
FIG. 2.—IRON MOLD AND TEST.

thereto. Both the absorption and delivery tube are rinsed with cold water and any adhering precipitate is dissolved by means of a fine jet of the dilute hydrochloric acid. Care is exercised that the acid solution be not unduly agitated before the introduction of the iodine solution. If the absorption is effected in the beaker the delivery tube is removed first, cleansed with the acid and water and, with the precaution suggested above, an excess of the dilute acid is added. The final volume of the solution should approximate 400 cc. The solution is stirred gently in order to neutralize the alkali of the absorbent, which, if left undisturbed, reacts with the iodine with formation of a substitution product from which the iodine is but partially liberated upon acidifying the solution. The iodine solution is then added in successive portions, while the solution is stirred

as the color fades, until the addition of a final drop yields the permanent blue color of the adopted endpoint. The number of cubic centimeters of the iodine required multiplied by .01 is equivalent to per cent sulphur.

The volume of the solution in the absorption vessels is so adjusted as to insure a sufficient body for the absorption of all of the gas, and a sufficient excess of the stock absorbent is maintained in proportion to the total volume of the solution. The proportions as used in the 10-in. by 1-in. test tube (15 cc. of any of the stock absorbents diluted to 60 cc.) may be taken as a basis therefor. If desired, the starch indicator solution may be added to any of the stock absorbent solutions in such proportions as to furnish the equivalent of 10 cc. of the indicator in the titration.

Solutions Required.

Starch Indicator.—To 1 litre of boiling water in a suitable flask, a cold emulsion of 6 grams of starch suspended in 100 cc. of cold water is cautiously added, and the boiling continued five minutes after the final addition. The solution is cooled to room temperature, and, with the addition of 6 grams of zinc chloride in 50 cc. of cold water, mixed thoroughly and set aside for a day or two with occasional shaking. After the final subsidence of the heavier residue the starch solution is siphoned into the container and the settlings rejected. The zinc chloride is added as a preservative. Preliminary boiling of the water tends to lessen the frothing or formation of large bubbles on the addition of the starch.

Standard Iodine.—Ten grams of potassium iodide crystals, followed by 45 grams of resublimed iodine, are transferred to a glass-stoppered graduated flask; 25 cc. of cold water are added and the flask is shaken until the iodine is completely dissolved. The solution is diluted to volume of 1 litre and set aside for not less than 24 hours before standardizing. The known sulphur of a standard steel, as determined both by the evolution and oxidation methods, is used as the basis of standardization with final adjustment so that 1 cc. is equivalent to .01 per cent sulphur on the basis of the weight of sample used.

Dilute Hydrochloric Acid

Concentrated Acid	500 cc.
Water	500 cc.

Absorbents.

Ammoniacal Solution of Cadmium Chloride

Cadmium Chloride	5 grams
Water	375 cc.
Ammonia	625 cc.

Caustic Alkali

Caustic Potash or Soda	5 grams
Water	1,000 cc.

Ammoniacal Solution of Zinc Sulphate

Zinc Sulphate	10 grams
Strong Ammonia	625 cc.
Water	375 cc.

Gravimetric Method.

Five grams of the sample are transferred to a clean No. 4 beaker or twelve ounce Erlenmeyer flask, and with the addition of 100 cc. of concentrated nitric acid, gentle heat applied until solution is complete. It is sometimes necessary to check a too violent action of the acid, which might result in loss of unoxidized sulphur, by placing the beaker or flask in cold water. The solution is boiled briskly to low bulk, and after cooling somewhat, 30 cc. of strong hydrochloric acid added, the solution is again concentrated, and following the precautionary addition of one gram of pure sodium carbonate, transferred to a clean porcelain dish and evaporated to complete dryness and the residue baked a short time at a moderate temperature.

The dish is allowed to cool, 35 cc. of concentrated hydrochloric acid added and heat applied until the soluble salts dissolve; the solution is again evaporated to complete dryness and

the residue baked as before. This procedure insures the elimination of the nitric acid. The dish is cooled, 35 cc. of concentrated hydrochloric acid are added and heat is applied until the soluble salts are again dissolved. The cover is removed, and with the dish placed upon a suitable asbestos ring in order to prevent the formation of a crust of the metallic salts upon the sides, the solution is evaporated to the first appearance of the separation of the chlorides, which are then dissolved by the addition of a few drops of concentrated hydrochloric acid; three drops should suffice. Cold water is added to volume of about 75 cc., the residue is collected upon an 11 cm. filter and the filtrate received in a No. 2 beaker. The residue and filter are washed with the least possible amount of the hot dilute hydrochloric acid, alternated with cold water, to the disappearance of the iron stains and finally with hot water to volume of about 200 cc. To this cold solution 10 cc. of a 10 per cent solution of barium chloride are added and the solution is allowed to stand with occasional stirring not less than twelve hours.

When the precipitate of barium sulphate has subsided, it is collected upon a 9 cm. close ashless filter, washed with dilute hydrochloric acid (5 per cent) to the disappearance of the iron and finally with hot water until free from chlorides. The filter and residue are ignited uncovered in a weighed crucible at a moderate temperature to the disappearance of the carbon of the paper and finally at a somewhat increased temperature for ten minutes. The residual weight multiplied by 13.74 and divided by 5 equals per cent sulphur. The filtrate is set aside until satisfied that no further precipitation ensues.

The carbonaceous siliceous residue of the original solution usually retains some sulphur and is ignited in platinum under cover of pure sodium carbonate and decomposed by fusion with further addition of the carbonate and two-tenths gram of nitre; the subsequent hydrochloric acid solution is freed from its silica and added to the original filtrate, or examined separately for sulphur.

It is apparent that absolute purity of reagents and cleanliness of utensils and the final weighing of the barium sulphate effected under the most exact conditions, are determining factors in the successful issue of analyses by this method.

Determination of Phosphorus.
Volumetric Method.

Acid Alkali Titration

One or two grams of the sample are transferred to a flask, preferably twelve ounce Erlenmeyer, 40 cc. of nitric acid (1.13 sp. gr.) added and heat applied until the sample is decomposed. The sides of the flask are washed down, one gram of ammonium persulphate is added and the solution boiled until the combined carbon is completely oxidized. The residue is caught on an 11 cm. filter and washed with dilute nitric acid (two per cent) and hot water to the disappearance of the soluble iron. The filtrate is heated to boiling in a suitable flask, a slight excess of a solution of potassium permanganate added and the boiling is continued until the excess of the permanganate is decomposed. The resultant precipitate of manganese dioxide is reduced to its soluble nitrate by means of the chosen reducing agent and the boiling continued until the excess thereof is decomposed.

With the solution at a temperature of 80 deg. C., 40 to 60 cc. of ammonium molybdate solution are introduced and the flask is shaken for five minutes. The precipitate quickly subsides and is at once collected upon a 9 cm. filter, washed with dilute nitric acid (two per cent) until free from iron, and then with solution of potassium nitrate (one per cent) to the removal of the last trace of free acid.

The titration may be conducted in a beaker or in the flask in which the precipitation was effected. Care is exercised in the latter case to wash the flask thoroughly with the neutral wash, as any acid present of course reacts with the standard alkali. The filter and precipitate are transferred to the beaker or flask and the standard alkali is added in excess sufficient to render

easy the subsequent maceration of the filter, which is accomplished by breaking up with a clean glass rod, or, if in a flask, by shaking the flask closed by a clean rubber stopper. When the precipitate has dissolved and the paper is thoroughly disintegrated, the solution is diluted and three drops of the indicator are added. The standard acid is then added to the disappearance of the pink tint, the sides of the titrating vessel washed down with cold water and, with a final volume of about 150 cc., the standard alkali is added to the adopted pink endpoint. The number of cubic centimeters of the alkali used in excess of the acid, multiplied by .01 or .02, is the equivalent of phosphorus in hundredths per cent.

Solutions Required.

Standard Caustic Soda.—Six and six-tenths grams of pure sodium hydroxide are dissolved in 100 cc. of boiled water. The solution is freed from any carbonic acid introduced by the caustic alkali, by the addition of about 1 cc. of a saturated solution of barium hydroxide with subsequent filtration, and the solution finally diluted to volume of one litre. If desired, a stock solution may be carried, a convenient aliquot portion of which is diluted for the working solution.

Standard Nitric Acid.—Ten cc. of concentrated acid diluted to one litre. The acid and alkali solutions are standardized against the known phosphorus of a standard steel with final correction so that they are exactly agreeing and 1 cc. is equivalent to .01 per cent phosphorus on the basis of two grams of the sample.

Ammonium Molybdate.—To 65 grams of molybdic acid (85 per cent) suspended in 142 cc. of cold water in a suitable flask, 143 cc. of strong ammonia are added and the flask is rotated until the molybdic acid is completely dissolved. The solution is cooled and added slowly to 715 cc. of nitric acid of 1.20 sp. gr. while the acid solution is vigorously stirred.

Molybdic acid is at times encountered which fails to yield a permanently clear solution of the ammonium molybdate; the addition of one or more drops of a 10 per cent solution of ammonium phosphate assists in clarifying the solution. The solution is allowed to stand over night, when it is filtered into the container, preferably through purified asbestos and with the aid of suction.

Phenolphthalein.—One gram dissolved in 1 litre of ethyl alcohol.

Nitric Acid (1.13 sp. gr.)

Nitric Acid—Concentrated .... 240 cc.
Water .... 760 cc.

Potassium Permanganate Oxidant.—Twenty-five grams to 1 litre.

Reducing Agents.

Ammonium Bisulphite.—Five per cent solution.

Ferrous Sulphate.

Ferrous Sulphate .... 50 grams
Water .... 1000 cc.

Potassium Nitrite.

Potassium Nitrite .... 50 grams
Water .... 1000 cc.

Sugar. Saturated solution.

Potassium Permanganate Titration.

The yellow precipitate as above obtained is washed with ammonium sulphate solution, or if desired, dilute sulphuric acid (2 per cent) until free from iron and molybdenum salts. Dilute ammonia (1 part ammonia and 3 parts water) is poured over the precipitate and the filter is washed with hot water, and the solution caught in the flask in which the precipitation was effected. The volume of the solution should not exceed 25 cc. Eight grams of granulated zinc, 20 mesh, followed by 75 cc. of dilute sulphuric acid are added and the solution is gently boiled until reduction is complete. The undissolved zinc is then collected upon a glass wool filter and both flask and filter are washed with cold water. To the cold filtrate, standard potassium permanganate solution is added until a slight per-

manent pink tint is obtained The measure of the permanganate solution required multiplied by its value in terms of phosphorus and divided by the initial weight of the sample equals per cent phosphorus.

Solutions Required.

Acid Ammonium Sulphate.

Strong Ammonia .... 15 cc.
Sulphuric Acid (1 part acid to 1 part water)...... 50 cc.
Diluted to 1 litre.

Standard Potassium Permanganate.—Two grams potassium permanganate dissolved in 1 litre of water. One cc. equals approximately .0056 per cent phosphorus on a 1 gram basis. The known phosphorus of a standard steel is used as the basis of standardization.

Gravimetric Method.

Weighing as Ammonium Phosphomolybdate.

One to five grams of the sample are transferred to a No. 5 glazed porcelain dish with cover glass, 25 to 60 cc. of nitric acid (1.13 sp. gr.) are cautiously added, gentle heat is applied until the sample is completely decomposed, and the solution is then rapidly boiled to dryness. The temperature is raised gradually with final heating of the uncovered dish over the full flame of an Argand or Bunsen burner to the complete expulsion of the acid. After cooling sufficiently to avoid danger of breaking the dish, 30 cc. of concentrated hydrochloric acid are added, heat is applied until the soluble salts are dissolved, and the solution rapidly boiled to the first appearance of the separation of the chlorides. This evaporation is conducted with the same regard to the prevention of the formation of a crust upon the sides of the dish as in the previously described method for the determination of sulphur. The dish is removed from the heat, 10 cc. of strong nitric acid added and heat again applied for about 1 minute. Cold water is added to volume of about 60 cc., the solution stirred with a clean glass rod and filtered into a suitable flask; the residue, collected upon an 11 cm. filter, is washed with 2 per cent nitric acid and hot water alternately until free from iron, which insures also the complete removal of the soluble phosphate.

When the iron contains titanium in amount sufficient to interfere with the precipitation of the phosphorus and an exact determination is required, the residue of silica and graphitic carbon is ignited in a platinum crucible, the silica volatilized with hydrofluoric acid in the presence of a few drops of sulphuric acid and the resultant residue ignited and fused with sodium carbonate. The melt is boiled with hot water until disintegrated, the insoluble titanate is collected upon a filter and the soluble phosphate added to the main solution.

To the clear filtrate with a volume of about 150 cc., 25 cc. of strong ammonia are added at once and the flask is shaken until the resultant precipitate has coagulated. The addition now of 25 cc. of strong nitric acid renders the solution clear and furnishes an excess of about 10 cc. of strong nitric acid—which, it is found, yields a pure crystalline precipitate of the phosphomolybdate with no tendency to creep. The relatively large amounts of ammonia and nitric acid added are prescribed in the case of 5 gram samples and are essential in that they insure a sufficient proportion of ammonium nitrate in the solution. It is evident that these additions may be decreased with smaller initial weights of the sample, but it will be found advantageous to maintain approximately the excess of 10 cc. of the nitric acid.

With the solution at a temperature of 80 deg. C., 50 to 75 cc. of the molybdate solution are introduced and the solution agitated for about five minutes. When the yellow precipitate has finally subsided, it is collected upon a previously dried and weighed 9 cm. filter, the filter and contents are washed thoroughly with 2 per cent nitric acid, dried for one hour in an air bath at a temperature of about 115 deg. C. and weighed between watch glasses with ground edges and enclosed in a suitable clip. It is of course essential that the light weight of the ....ly measure...

previously dried paper shall have been obtained under the same conditions. The increase in weight multiplied by 1.63 and divided by the initial weight taken is equivalent to per cent phosphorus.

Weighing as Magnesium Pyrophosphate.

The yellow precipitate as above obtained is dissolved on the filter with hot ammonium citrate solution and the filter thoroughly washed with hot water, and the solution retained in a No. 0 beaker. To this cold solution, 5 to 10 cc. of the magnesia mixture are added, the solution is well stirred and allowed to stand in the cold not less than three hours. The precipitate of magnesium phosphate is collected upon a 9 cm. close ashless filter, washed with dilute ammonia, ignited at a low temperature to the disappearance of the carbon of the filter and finally for ten minutes at the full temperature of the blast. The weight of the magnesium pyrophosphate multiplied by 27.87 and divided by the initial weight taken is equivalent to per cent phosphorus.

Solutions Required.

Ammonium Citrate.
Citric Acid	50 grams
Strong Ammonia	350 cc.
Water	650 cc.

Magnesia Mixture.
Magnesium Sulphate	84 grams
Ammonium Chloride	250 grams
Water	665 cc.
Ammonia	335 cc.

Dilute Ammonia.
Strong Ammonia	200 cc.
Water	800 cc.

Determination of Manganese.
Walters' Color Comparison Method.

Fifty to two hundred mg. of the sample are transferred to a suitable tube or flask, preferably an 8-in. by 1-in. test tube or a 2-oz. Erlenmeyer flask, 10 cc. of nitric acid (1.20 sp. gr.) added and heat applied either in a water bath or on a hot plate until the sample is decomposed. About one-half gram of moist ammonium persulphate, or its equivalent solution, is added and the heating continued until the combined carbon is completely oxidized. If silica or graphitic carbon separate in amount sufficient to render it necessary, the solution is filtered into a similar vessel and the residue retained on a small filter. The filter and residue are washed with the minimum of water, or the 15 cc. of silver nitrate solution required for the subsequent oxidation. The silver nitrate is introduced either through the filter or into the unfiltered solution, one gram of moist persulphate is added and the heating continued until the characteristic pink tint of the permanganic acid is well defined. The solution is cooled, transferred to the comparison tube and the color matched against a suitable standard which has been treated in exactly the same manner and with the same relative initial weight. The relative volume of the solution as required to match the color of the standard is equivalent to hundredths per cent manganese.

Volumetric Method.

The solution of permanganic acid is transferred to a beaker or flask, and following the addition of 10 cc. of sodium chloride solution, the standard sodium arsenite is added to the definite end point of the disappearance of the pink tint. The volume of the solution should not exceed 75 cc. If the silver is not precipitated, the reaction of the persulphate with the silver nitrate renders the endpoint somewhat fleeting and recurring. If conducted without delay, however, the titration may be accomplished with perfect satisfaction. The end point in this case while not so permanent is perhaps somewhat sharper than in the presence of the chloride.

When desired, a greater initial weight of the sample may be taken, the solution thereof diluted to a definite volume, the

graphite and silica allowed to subside and aliquot portions withdrawn with a pipette.

Solutions Required.

Sodium Chloride.—Two and one-half grams sodium chloride to 1 litre of water.

Silver Nitrate.—About 66.5 grams of silver nitrate are dissolved in 1 litre of water. Twenty cc. of this diluted to 1 litre affords a solution of which 15 cc. is equivalent to approximately .02 gram of silver nitrate.

Sodium Arsenite.—A stock solution is prepared by dissolving 10 grams of C. P. arsenious acid in a boiling aqueous solution of sodium carbonate (30 grams dissolved in 250 cc. of water), with final dilution of the cold solution to volume of 1 litre. Sixty-two and one-half cc. of this solution are diluted to volume of 1 litre for the working solution of which each cc. is equivalent to approximately 1/10 per cent manganese, with an initial weight of 2/10 gram of the sample. The known manganese of a standard steel or iron is used as the basis of standardization.

Ford's Gravimetric Method.

Five grams of the sample are transferred to a No. 5 glazed porcelain dish with cover glass, 75 cc. of nitric acid (1.20 sp. gr.) cautiously added and gentle heat applied until the sample is decomposed, and the solution then rapidly boiled to complete dryness. The residue is baked a short time at a moderate temperature. The dish is cooled, 35 cc. of strong hydrochloric acid are added and heat is applied until the soluble salts dissolve. Cold water is added to approximately 60 cc., the solution is filtered into a No. 5 beaker and the residue caught on an 11 cm. filter and washed with the least amount necessary of hot dilute hydrochloric acid and cold water alternately, to the disappearance of the soluble metallic salts. The filtrate is boiled to low bulk, cooled, 50 cc. of strong nitric acid are added and the solution is again boiled down to low bulk. The evaporation is repeated with successive portions of nitric acid to the complete expulsion of the hydrochloric acid and with conversion of the chlorides to their corresponding nitrates; the evaporation at this point is carried to the first indication of the separation of the nitrates as observed in the formation of a small nucleus of scum floating on the surface of the solution. After the solution has cooled somewhat, 75 cc. of strong nitric acid are added and the solution is again brought to the boiling point. To the boiling solution, crystals of potassium chlorate (five grams will be found sufficient) are cautiously added in small portions from a glass spoon, and the solution is boiled for five minutes.

The solution is allowed to cool before filtering. Filtration is effected through the medium of an asbestos plug held in a carbon filtering tube. The asbestos is prepared for use by heating with aqua regia to remove soluble lime salts and the plug is washed with strong nitric acid before using. The precipitated manganese dioxide on the filter is then washed with successive additions of strong nitric acid (free from nitrous fumes) to the practical elimination of the iron; the acid is added first to the beaker and transferred from thence to the filter. After the final washing, suction is continued until the plug is practically dry, when the rubber stopper carrying the filter is transferred to a clean flask. The filtering tube is filled with one per cent solution of ammonium bisulphite and allowed to stand about one minute, suction is then applied, successive portions of the bisulphite solution being added until the last trace of manganese dioxide is dissolved; the soluble manganese salt is finally washed from the asbestos filter with hot water. The solution is transferred to a No. 4 beaker or, if preferred, retained in the flask, 10 cc. of strong nitric acid added and the excess of the bisulphite decomposed by boiling. The manganese dioxide could be dissolved perhaps more conveniently by means of hot hydrochloric acid, but the use of the bisulphite solution obviates the possible danger of contamination by traces of soluble lime or

magnesia, which in some grades of asbestos seem to be in evidence notwithstanding numerous successive treatments with the hot acid.

The solution is cooled somewhat and ammonia is added to the point of a faint permanent precipitate, followed by the addition of 15 cc. of a 20 per cent solution of ammonium acetate. The solution is boiled two minutes, filtered into a No. 5 beaker and the precipitate of the basic acetate of iron collected on an 11 cm. filter. The beaker or flask and the filter are washed twice with hot water, the precipitate is dissolved with hot dilute nitric acid and the solution returned to the vessel in which the precipitation was effected. The filter is washed with hot water until free from iron and the basic acetate precipitation and subsequent filtration repeated.

To the combined filtrates 5 cc. of glacial acetic acid are added and the solution is again brought to the boiling point; the cover glass is removed, 10 cc. of a 10 per cent solution of ammonium or sodium phosphate added, and with the solution still at the boiling point and with constant stirring, strong ammonia is added to the first appearance of an incipient opalescence. The stirring is continued until the opalescence has developed into the characteristic crystalline precipitate, at which point the addition of the ammonia is continued, drop by drop and with constant stirring, until the total of 25 cc. has been added.

The precipitate is filtered at once upon an 11 cm. ashless filter and washed with dilute ammonia water (2 per cent) until free from soluble salts. The filter and residue are ignited in a weighed porcelain or alundum crucible at a low temperature until the paper is burned off, and finally at a somewhat higher but still moderate temperature for ten minutes.

$$\frac{\text{Residual weight} \times 38.69}{5} = \text{Per Cent Manganese.}$$

Determination of Total Carbon.
Direct Combustion Method.

The direct combustion of the carbon of iron or steel and their various alloys is fast coming to supersede the more tedious wet method. Certain limitations of this most excellent method, however, are to be noted. The inherent difficulty of successfully burning particles of steel of too great mass is apparent. Certain samples of both iron and steel or their alloys have been encountered of such a character as to necessitate the intimate admixture therewith of a finely divided metallic oxide, or, in the case of certain irons, fine drillings of low carbon steel to assist in the oxidation. The powder of the crushed test of white iron requires a somewhat longer time for complete combustion than the drillings from the test of gray iron but with proper care may be burned with perfect satisfaction. An essential requirement in the conditions of the method is the close control of the temperature which should approximate a constant of 1000 deg. C. and should not fall below 960 deg. C. A pyrometer couple will be found convenient in controlling the temperature.

The combustion is conducted preferably in a fused quartz tube in either an electric or suitable gas-fired furnace. A platinum tube may be used but is subject to damage from possible splashing of the oxidized metal and is, moreover, much more expensive than the quartz. The use of litharge or other volatile and easily reducible oxides is, of course, prohibited in the platinum tube. The quartz tube is 24 inches long by three-quarters inch or seven-eighths inch inside diameter, free from seams and selected of uniform diameter. Platinized asbestos, which may be secured at either end by short plugs of platinum gauze, is inserted for a space of five or six inches in the tube immediately following the position to be occupied by the boat. The rubber stopper at the end of the tube may be protected by the insertion of a removable plug of asbestos conveniently enclosed in a thin roll of platinum gauze. Platinum, nickel, alundum, porcelain or clay boats may be used. The dimensions, 3 to 3½ in. long, by ¼ in. deep, ½ in. wide at the bottom, flar-

ing to ⅝ in. at the top, all inside measurements, will be found convenient. Suitable furnaces for either direct or alternating current, with transformers, rheostats, etc., to suit the individual requirements may be obtained of the trade.

The oxygen from the high pressure cylinder under control of the reducing valve is purified by passing through caustic potash solution of 1.27 sp. gr. contained in washing bottles, preferably two arranged in tandem. A glass tube of convenient form and size, filled with granulated zinc, may be connected to the exit end of the combustion tube to retain possible fumes of sulphuric anhydride formed in the combustion of high sulphur irons. When the potash absorbent is used, a calcium chloride drying tube preceding the bulb or bottle is necessary.

Factor weights may conveniently be used in the direct combustion as follows: For the potash absorption and weighing of the bulb, .2727 gram or .5454 gram, where each one-tenth mg. gain in weight of the bulb is equivalent to .01 per cent or .005 per cent carbon respectively. For the barium hydrate absorption, with ignition of the barium carbonate, .304 gram or .608 gram, where each one-tenth mg. of the weight of barium carbonate is equivalent to .002 per cent and .001 per cent carbon respectively. For the titration methods it is convenient to employ for the initial weight even gram multiples, preferably 250 mg. to 500 mg.

One-quarter or one-half gram or the suitable factor weight of the sample is transferred to the boat. The boat is partially filled with ignited alundum depressed in the center in the form of a V about 1 in. in length, and in which the sample is spread in a compact mass. This intimate contact of the particles of iron conserves the heat for the continuous combustion of the carbon. The weighed potash container or the Meyer bulb tube, to which has been added 80-100 cc. of the barium hydroxide solution or 50 cc. of the standard potash solution, is connected with the tube and the boat is carefully pushed to its position in the center of the heat zone. The stopper of the tube is replaced and, with all connections tight, the current of oxygen is started at a rate approximating a continuous bubbling, as observed in the purifying solutions. It is essential that the tube should have acquired the maximum temperature before the introduction of the boat, a condition which will naturally obtain in the case of a continuous routine of combustion.

The first indication of the oxidation is observed in the sudden decrease of the bubbling of the oxygen through the absorption vessel. As the rate decreases the flow of oxygen is somewhat accelerated until the conclusion of the oxidation process as indicated by the recurrence of the rapid flow through the exit; the flow of oxygen is then reduced to the normal rate which is maintained for ten minutes to insure the complete oxidation of all the carbon and the final absorption of the resultant carbon dioxide. The absorption vessel is disconnected, the current of oxygen discontinued and the analysis concluded by either weighing the potash bulb; titrating the standard solution of caustic potash; filtering, igniting and weighing the precipitated barium carbonate; or titrating the excess of the barium hydrate solution with a weak solution of hydrochloric acid and without preliminary filtration.

By Increase of the Weight of the Potash Bulb or Bottle.

The perfectly clean and dry bulb or bottle is at once weighed. The increase in weight as noted in connection with the initial weight of the sample is corrected to per cent carbon.

The potash container is prepared for use by carefully introducing the required volume of the solution, connecting with the apparatus and passing a current of oxygen through the hot combustion tube to the complete displacement of the air in the container. The vessel is disconnected and at once weighed. The weight so obtained furnishes the light weight for the succeeding combustion.

Titration of the Caustic Alkali.

The absorption in this case is effected in a Meyer bulb tube containing 50 cc. of the potash solution, accurately measured.

The contents of the tube are transferred to a flask, the tube is cleansed with freshly boiled water, three drops of the phenolphthalein solution are introduced and dilute hydrochloric acid (one part acid to one part water) carefully added until the color begins to fade. The more dilute standard acid is then added until the addition of one drop just discharges the pink tint. The excess alkali which has not entered into combination with the carbon dioxide has now been neutralized and the solution is ready for the actual titration. One drop of the methyl orange or methyl red solution is added and the standard acid added in successive portions until the addition of one drop causes the characteristic change from the yellow to the rose tint in the solution. The measure of the acid required for this titration minus the equivalent required in the blank titration and multiplied by the carbon value of the solution equals per cent carbon.

If preferred, the titration may be conducted with the phenolphthalein indicator, the measure of the standard acid required to discharge the pink tint being deducted from the blank equivalent and calculated to per cent carbon as above. This method, however, would not appear to offer any advantage over the other and obviously requires more of the standard acid.

The blank equivalent is established by passing the oxygen through the hot combustion tube and into the measured volume of the absorbent solution with final titration as outlined above. The carbon value of the acid solution is established by titration of the measured volume of the potash solution following the combustion of a standard steel. Additional confirmation may be had by titrating against a standard solution of sodium carbonate. The use of sodium oxalate in this connection is recommended.

Solutions Required.

Standard Acid, one-sixth normal. Fifteen cc. hydrochloric acid (1.20 sp. gr.) diluted to 1 litre.

Caustic Potash, one-sixth normal. Nine and one-half grams diluted to 1 litre. This solution need not of necessity be absolutely one-sixth normal.

Phenolphthalein. One gram dissolved in one litre of ethyl alcohol.

Methyl Orange. One gram dissolved in 1 litre of water.

Methyl Red. One gram dissolved in 1 litre of ethyl alcohol.

Filtration and Ignition of the Barium Carbonate.

The bulb tube is detached from the train, the index finger of the left hand being simultaneously placed over the exit end in order to prevent the solution from flowing back in the tube. The solution is filtered through an 11 cm. ashless filter, the filter and precipitate are washed with freshly boiled water (six good washings will be found sufficient), ignited in a weighed platinum crucible at a moderate temperature to the disappearance of the carbon of the filter and finally at a good full heat for five minutes. The weight of the barium carbonate multiplied by 6.08 and divided by the weight taken, or with correction in agreement with the initial factor weight, is equivalent to per cent carbon. The solution and precipitate are transferred to the filter either by way of the bulb entrance with control of the flow by the index finger upon the open exit end of the tube or by control with the thumb placed over the bulb entrance with the solution poured through the exit end. The precipitate may be readily removed by shaking the tube after successive additions of the water. The washing is conducted as thoroughly and expeditiously as possible and care is exercised to avoid the passage of air unduly through the filter.

Titration of the Excess Barium Hydroxide.

The bulb tube is detached as before, the solution and precipitate are transferred to a 12-ounce Erlenmeyer flask, and the tube thoroughly washed with freshly boiled water. Three drops of the indicator solution are introduced and the standard hydrochloric acid is added in successive small portions, while the flask is shaken, to the disappearance of the pink tint. The agi-

tation of the solution insures the recovery of any carbon dioxide liberated upon the addition of the acid. The equivalent of the acid required in a preliminary blank titration minus the measure required in the actual titration, multiplied by the factor of the solution as determined by frequent combustions of a standard steel and in agreement with the initial weight of the sample, is equivalent to per cent carbon.

Solutions Required.

Standard Hydrochloric Acid.—Eight cc. of strong hydrochloric acid diluted to 1 litre. Freshly boiled water should be used in making up the solution. One cc. of this solution is equivalent to approximately .05 per cent carbon on a 1 gram basis.

Barium Hydroxide.—Twenty grams of barium hydroxide crystals are dissolved in 1 litre of hot freshly boiled water and the flask covered and set aside to cool. When the solution has reached the room temperature, it is filtered as rapidly as possible through a 25 cm. filter into the container. The solution is preferably withdrawn from the container by means of an overflow pipette in conjunction with a suitable guard tube filled with soda lime or caustic potash solution.

Phenolphthalein. One gram dissolved in 1 litre of ethyl alcohol.

Solution and Combustion Method.

The conditions involved in the operation of this method admit of the employment of a greater initial weight of sample than in the previously described Direct Combustion Method.

One gram, or if the barium hydroxide absorbent is to be used, a ten factor weight (.608 gram) of the sample is transferred to a No. 2 beaker or 4-oz. Erlenmeyer flask; 100 cc. of the solution of copper and ammonium chloride or the potassium salt are added and the covered beaker is placed on a suitable stirring machine or stirred by hand, at a temperature of 60 to 70 deg. C. until in solution; or, if a flask is used, it is placed on a shaking machine or closed with a clean rubber stopper and shaken by hand until solution is complete.

The carbonaceous residue is collected upon a filter of purified ignited asbestos made up in a perforated platinum boat, Gooch crucible or other suitable form of holder. The residue is washed by decantation with dilute hydrochloric acid to the disappearance of color in the washings, then transferred to the filter and washed with cold water until the acid is completely removed. Preliminary drying of the residue is not essential in routine work, but where the refinement of accuracy is desired, it is recommended that the residue be dried in a hot air or steam bath at a temperature of 100 deg. to 105 deg. C. The residue is ignited in a platinum or silica tube, combustion crucible or other form of apparatus and in an atmosphere of pure oxygen or air free from carbon dioxide. The apparatus and the subsequent treatment of this carbonaceous residue may conveniently conform to the description of the method by direct combustion, and the final measure of the carbon established by any of the several optional methods therein described.

Solutions Required.

Copper and Ammonium Chloride.—Three hundred grams of the pure salt are dissolved in 1 litre of boiling water. After cooling, 60 cc. of strong hydrochloric acid are added and the solution is filtered through a purified asbestos plug into the container.

Copper and Potassium Chloride.—One pound dissolved in 1 litre of water, as above, with subsequent similar treatment.

Caustic Potash. (1.27 sp. gr).—Three hundred grams dissolved in 1 litre of water.

Determination of Graphitic Carbon.

One gram or the ten factor weight (.608 gram) is transferred to a No. 2 beaker or a 4-oz. Erlenmeyer flask, 40 cc. of nitric acid (1.13 sp. gr.) added and gentle heat applied until solution is complete. The solution is then boiled for a few minutes, the

residue is collected upon a suitable asbestos or paper filter, washed alternately with hot dilute hydrochloric acid and water until free from iron and finally with hot water until free from acid. If the filtration has been conducted through asbestos filter, the residue may be washed with hot caustic potash solution (1.10 sp. gr.) to remove separated silica. If preferred, a few drops of hydrofluoric acid may be added to the acid solution to dissolve the silica which might otherwise interfere with the filtration.

The analysis may be concluded in either of the following optional ways:

First. The graphitic residue is collected upon a suitable filter, washed with dilute hydrochloric acid and water and ignited in the combustion apparatus, and the measure of the graphitic content established as in the case of total carbon. It will be found convenient in this case to use the ten factor weight of the sample.

Second. The filter and graphitic residue are placed in a crucible and dried to constant weight at a temperature of 110 deg. C., then ignited until the graphitic carbon and the filter paper are completely oxidized and again weighed. The loss in weight minus the weight of the filter paper when used and multiplied by one hundred is equivalent to per cent graphitic carbon on a 1-gram sample.

Determination of Combined Carbon.
Indirect Method.

The per cent graphitic carbon as found above subtracted from that of the total yields the measure of the combined carbon. If desired, a direct estimation of the combined carbon may be conducted by the color comparison method.

Color Comparison Method.

The relatively exact chemical characteristics and preliminary heat treatment of the test piece and standard are universally recognized as fundamental bases of accuracy in the determination of the combined carbon of iron or steel by the color comparison method. The universal and continued use of this most convenient method in the analysis of steel products attests its efficiency within the close limits of control possible of attainment in the preparation of the steel standards. The application of the method to the determination of the combined carbon of the various grades of iron is much more limited owing to the larger and varying content of manganese and sulphur, the principal disturbing chemical factors affecting the color of the solution to be compared. It is apparent, however, that with a judicious selection of standards and careful control of all factors involved, the color comparison method may be applied to the determination of the combined carbon content of iron with a quite reasonable degree of accuracy.

The method of procedure as recommended is as follows: One gram of the sample is transferred to a dry clean beaker, flask or tube, as desired. In the presence of graphitic carbon, the use of a magnetized spatula or wire in weighing off the sample is, of course, prohibited. Thirty cc. of nitric acid (1.20 sp. gr.) are added and heat is applied until the sample is decomposed. Graphitic carbon and silica are removed by filtration, and the filtrate retained in a 100 cc. graduated flask or cylinder. The filter is washed with cold water to the disappearance of color; the solution is diluted to volume, thoroughly mixed, and 10 cc. withdrawn with a pipette and transferred to the comparison tube. The color is then compared with the color of the solution of a standard iron in a similar tube and which has been treated in exactly the same manner and with the same relative initial weight. The simple calculation involved in the relative volumes of the two solutions yields the per cent carbon in the sample.

A convenient aliquot portion of the above solution may be employed for the determination of manganese by the persulphate color method.

Determination of Titanium.

Five grams of the sample are transferred to a No. 5 glazed porcelain dish or to a No. 4 beaker, with cover glass; 50 cc. of strong hydrochloric acid are added and the sample is digested until completely decomposed. Practically all of the titanium is said to remain insoluble with the silica and graphitic carbon. The insoluble residue is collected upon an 11 cm. filter, washed a few times with hot water, ignited in a platinum crucible and the silica volatilized with hydrofluoric acid in the presence of a few drops of sulphuric acid. The residue is then carefully ignited to the complete expulsion of the hydrofluoric acid, which if retained prohibits the subsequent colorimetric determination of the titanium.

Should it be desired to determine the usually quite negligible amount of titanium in the filtrate from the insoluble residue, it is recovered as follows: The filtrate in a No. 5 beaker is diluted to 250 cc. and strong ammonia added to the first appearance of a precipitate which slowly dissolves upon stirring. Dilute ammonia (2 per cent) is added to a faint permanent precipitate, which is then dissolved by the addition of 15 cc. of a 10 per cent solution of hydrochloric acid. The precipitate thus formed should dissolve rather slowly following vigorous stirring of the solution. One hundred cc. or more of a 20 per cent solution of sodium hyposulphite are added and the solution in the beaker is stirred until the iron is completely reduced and the free sulphur begins to separate. The solution is then boiled for ten minutes and the precipitated titanic acid allowed to subside and collected upon an 11 cm. filter. The filter and precipitate are washed free from soluble salts with dilute acetic acid (2 per cent) and ignited in the crucible containing the ignited residue from the previously conducted volatilization.

The combined residues are fused with 4 grams of sodium carbonate, the melt is disintegrated by boiling with hot water, and the insoluble sodium titanate is collected upon a 9 cm. filter and washed with hot water containing a little sodium carbonate. Hot dilute sulphuric acid is added to the crucible which is then heated until the titanate is completely dissolved. The filter is spread upon the bottom of a No. 2 beaker, the contents of the crucible transferred thereto, and the crucible washed thoroughly with hot water. The beaker is then heated until the titanium salt dissolves, when the paper is removed and its burden of titanium sulphate returned to the beaker by washing with hot water. The solution is then transferred to the color comparison tube or if necessary to a graduated flask and aliquot portions taken. Shreds of filter paper which may be observed in the solution are removed by filtration.

In the color comparison, 5 cc. of C. P. hydrogen peroxide are added to the solution in the comparison tube and the solution is diluted to a volume yielding a color convenient for the subsequent comparison. A color equivalent to .005 gram of titanium in 100 cc. of solution will be found convenient. Water acidified with dilute sulphuric acid (equal parts acid and water) is then added to the standard tube to a volume allowing of subsequent necessary increase. 5 cc. of the hydrogen peroxide is added and the standard titanium solution added from a graduated burette in successive small additions until the colors agree. The measure of the standard solution required multiplied by its value in terms of titanium and corrected to agreement with the initial volume and weight is equivalent to per cent titanium. If the color of the solution yielded by a convenient weight of the sample is of insufficient depth for an accurate comparison, adjustment may be made as follows: A measured volume of the standard titanium solution, sufficient to furnish the required depth of color is added to the solution being tested and the comparison conducted as above described. The volume of the standard solution required in the comparison minus the equivalent previously added to the test solution is calculated to per cent titanium. The volume of the two solutions must, of course, be identical in the final comparison.

In preparing the standard titanium solution 1.05 grams of ig-

nited C. P. titanic acid are fused with 10 grams of sodium carbonate. The melt is digested with 100 cc. of hot water to the complete solution of the soluble alkali which is then filtered from the insoluble titanate. The residue of sodium titanate is washed with hot water containing a little sodium carbonate and dissolved from the filter with 100 cc. of dilute sulphuric acid (equal parts acid and water) and the solution diluted to 1 litre. The titanium equivalent of this solution is ascertained by igniting and weighing the precipitated titanic acid of an aliquot portion following a separation with sodium hyposulphite as previously described. The ignited salt will be found to contain approximately 95.6 per cent titanic acid. One cc. of the solution in the above proportion will therefore contain 0.6 mg. of titanium.

A gravimetric determination of the titanium in the iron may be conducted in the same manner, but inasmuch as in the colorimetric method the titanic acid as separated need be only relatively free from contamination by occluded salts, it is apparent that the latter method may be used to advantage, especially where many such determinations are required.

Determination of Copper.
Gravimetric Method.

Five grams of the sample are transferred to a glazed porcelain dish, 65 cc. of nitric acid (1.20 sp. gr.) are added and the covered dish is heated until solution is practically complete. Twenty-five cc. of dilute sulphuric acid (equal parts acid and water) are then added and the solution is evaporated to fumes of sulphuric anhydride. Ten cc. of the dilute sulphuric acid followed by 50 cc. of hot water are added, the residue is heated until in solution, and the solution is filtered into a No. 4 beaker and the filter washed with hot water. The filtrate is diluted to 300 cc., heated to boiling and, with the addition of 25 cc. of concentrated ammonium bisulphite solution, the boiling continued to the complete reduction of the iron. Thirty cc. of a 20 per cent solution of sodium hyposulphite are then added and the boiling is continued for five minutes or until the precipitated copper sulphide has coagulated. The precipitate is collected upon an ashless 11 cm. filter, washed with dilute nitric acid (2 per cent) and hot water, carefully ignited and weighed. The weight of the copper oxide as noted, multiplied by sixteen (80 divided by 5) equals per cent copper. Too great a temperature in the ignition causes the copper oxide to fuse on the crucible. As the copper oxide is somewhat hygroscopic, as are also some porcelain crucibles, less error is encountered by igniting in platinum.

The ignited copper oxide carries with it traces of contaminating iron. If it be desired to correct this usually negligible error, the impure oxide is dissolved in the crucible with 10 cc. of strong nitric acid, the solution transferred to a No. 2 beaker and diluted to 100 cc.; the iron is then removed by an ammonia precipitation. The precipitate so obtained is collected upon a 9 cm. filter, washed with hot water, ignited in the crucible in which the previous ignition was conducted and the weight of ferric oxide noted. This weight is deducted from the weight of the impure copper oxide and the correction applied.

Volumetric Method.

The nitric acid solution of the impure copper oxide in the No. 2 beaker obtained as above described, is rapidly evaporated to the removal of the bulk of the free acid, and the solution diluted with cold water to volume of 150 cc. Strong ammonia is then added from a burette until the addition of a final drop causes the permanent bluing of the solution, after which there is added by means of a pipette six drops of acetic acid. Ten cc. of a 40 per cent solution of potassium iodide are added and the standard sodium hyposulphite solution is run in until the color of the liberated iodine has almost disappeared. Ten cc. of the starch solution are now added and the addition of the standard sodium hyposulphite solution is continued to the final disappearance of the blue. The number of cubic centimeters of the standard solution required multiplied by its value in terms of copper and divided by five equals per cent copper in the sample.

The value of the standard hypo solution is determined by titration of a solution of pure copper or by titrating the solution of a steel of known copper content. The apparent large excess of the potassium iodide employed is because of its relation to the final end point in titration, the sharpness thereof decreasing in proportion as the excess of the iodide is decreased; the presence of ammonium salts in the solution also tends to lessen the permanency of the end point, hence the necessity for the preliminary evaporation of the acid solution.

If necessary, the time required for the volumetric determination may be materially lessened by initial solution of the sample in 50 cc. of dilute sulphuric acid (one part acid to three parts water), and the evaporation to dryness and filtration from the silicious residue omitted. The iron in this case not being oxidized, the boiling with the ammonium bisulphite is also omitted. The copper oxide so obtained is contaminated to a greater degree by occluded iron, which if it be present in the final titration is prejudicial to a satisfactory end point. This iron is, however, readily removed by filtration following the evaporation of the nitric acid solution previous to the addition of the ammonia. It is apparent also, that following the nitric acid solution of the sample, the preliminary treatment with ammonium bisulphite in the gravimetric method may be dispensed with, as the reduction may be accomplished by the addition of a sufficient excess of the sodium hyposulphite solution at the time of the precipitation of the copper sulphide. The precipitate formed under such conditions, however, carries an unduly large burden of separated sulphur, the presence of which would appear undesirable.

Solutions Required.

Standard Sodium Hyposulphite.—Eight grams to 1 litre. One cc. is equivalent to approximately .002 gram copper.

Starch Solution.—The solution as used in the volumetric determination of sulphur.

Multiple-deck slime concentrators designed by A. R. Wilfley are being used in some of the large copper mills in place of vanners. The machine consists of a series of concentrating decks, covered with grooved linoleum, and placed above each other. The space between decks is 1¼ in. The even distribution of the ore-pulp and wash water to the several decks is effected by an ingenious device, and is automatic. The table receives a simple longitudinal motion from an eccentric. The cycle of operation is divided into three periods: (1) feeding slime onto the decks, which are in a practically horizontal position. This continues for some minutes, during which the mineral becomes concentrated in the grooves. (2) Passing wash water onto the decks, which have now been tilted at a slight angle, to facilitate the removal of the upper layer of gangue which covers the concentrate. (3) Washing the concentrate out of the grooves and off the decks by means of a sudden flow of water. In this period the decks are tilted at a still greater angle. After a completion of the cycle, the table is returned to a horizontal position, ready for another feed of slime. Twelve decks are thus combined, giving a concentrating space of 864 sq. ft. The capacity of the machine is regarded to be equal to that of 8 vanners, and it requires about the same power and occupies the same space.

Nevada Consolidated Copper Company produced in the second quarter of this calendar year 18,092,439 pounds of copper. There was milled 813,141 tons of ore containing 1.66 per cent copper.

The United States Geological Survey has completed its final compilations showing metal productions in the various western states during 1911. Copies of the work can be had on application to the director, Washington, D. C.

Meeting of Lake Superior Mining Institute

The seventeenth annual meeting of the Lake Superior Mining Institute was held this year in the copper country, with headquarters at Houghton, Mich., on August 28, 29 and 30. In point of interesting papers presented, and large attendance from the copper and iron districts of Michigan and Minnesota, this session of the Institute was one of the most successful in its history. Before the meetings were over nearly 250 members and guests had registered with Secretary A. J. Yungbluth.

Many of the papers had been printed in advance and were distributed before the meeting. The visitors were provided further with a souvenir booklet, compiled by Mr. Arthur L. Carnahan, giving concise general information regarding the copper country, together with a map of the district and a tabulated statement of the mining and milling equipment of the principal copper companies.

Wednesday, August 28.

An afternoon excursion took the visitors first to the Michigan Smelting Works, where they witnessed the tapping of a reverberatory furnace and the casting of copper into various merchantable shapes. A Walker casting wheel is in use here, being the only one at present in the district.

Proceeding by train to the mines of the Copper Range Consolidated Company, the party inspected the power plant, blacksmith and machine shops of the Champion mine. At the Baltic mine interest centered chiefly in the dry and change house and in the sanitary provision which the company has made for the comfort and cleanliness of the men when they come off shift. Although excellent bathing facilities have been provided, the foreign laborers have been slow to take advantage of them; but the company believes that they will gradually become educated to use the baths more frequently and regularly, and ultimately appreciate this provision for their health and comfort.

The concrete oil-house at the Baltic mine is worthy of mention, as it represents an effort to systematize the storage and distribution of lubricants. Economical lubrication is a matter of importance at a large plant, as will be shown later in an abstract of a paper on that subject presented at one of the Institute meetings.

Returning to Houghton, the Institute convened in the evening for the reading and discussion of papers. Most of them related to mining and geology, and showed the keen interest which is being taken in the application of scientific principles to the solution of practical problems. The possible future extension of the copper-bearing area was discussed by Dr. A. C. Lane, of Tufts College, Mass. The proper location of shafts in relation to the copper lodes was presented by Dr. L. L. Hubbard, of Houghton. Other interesting papers presented were: Methods of Sampling at Lake Superior Iron Mines, by Benedict Crowell, of Cleveland, O.; System of Safety Inspection of the Cleveland-Cliffs Iron Company, by William Conibear, of Ishpeming, Mich.; Raising Shaft at Rolling Mill Mine, by E. N. Cory, of Negaunee, Mich.; Mine Sanitatioin, by E. B. Wilson, of Scranton, Pa.; Raising, Sinking and Concreting No. 3 Shaft, Negaunee Mine, by Mr. S. R. Elliott, of Negaunee, Michigan.

Economical Lubrication was the title of a paper prepared by **W. W. Davis,** of Boston, in which the author expressed the opinion that in these days of close attention to operating costs it has been found advisable to give greater care to the selection, storage, distribution and use of lubricants used in any plant of machinery, and especially in connection with large mining, milling and power companies. The paper first discusses the kinds of lubricants and their proper sphere of usefulness, and then takes up the methods necessary to secure economical lubricating costs.

"The selection of proper lubricants depends, of course, upon the class of machinery on which they are to be used. If on light-running and high-speed machinery the light-bodied or more fluid oils will give best results. For slow-speed machinery the heavier bodied oils will be better. For use on slow-speed engines where the oil is fed from cups a heavy-bodied oil should be used. For high-speed work and engines where continuous oiling systems are in use, a light-bodied oil should be used. Cylinder oils have for their base what is known, in the oil trade, as cylinder stock, of which there are two classes, the light colored or filtered stock and the dark or steam-refined stock, the latter being almost universally used. This cylinder stock is high in flash test and viscosity, but of itself would not make a good steam cylinder lubricant under ordinary saturated steam conditions; being a petroleum product, it has no affinity for moisture and will not stick to the wet cylinder and valve surfaces, so it is customary to compound this petroleum stock with a certain amount of fatty oil, usually tallow oil or neatsfoot oil, in order that it will emulsify with the steam and cling to the surfaces. Where the steam is fairly dry, 3 to 6 per cent is usually sufficient. Where the plant is small and the lubricating cost is a small matter, the most satisfactory method of purchasing lubricants would be to buy of some reliable oil firm, but where the cost of lubrication runs into thousands of dollars, as is the case with many corporations, and it is desired to obtain suitable lubricants at the lowest market price, the best plan is to purchase on specifications, stating clearly just what is needed, and awarding a contract for a year's supply to the lowest responsible bidder. Specification buying is fair both to the consumer and the dealer."

Service tests may be made on different oils by noting the rise in temperature in any bearing on a machine which is running under constant load and speed. Thermometers placed in the bearing and in the room near the machine will make it possible to secure data showing the temperature due to friction, and by plotting the results obtained with different lubricants an idea of their efficiency will be gained. In general, it is safe to assume that, of two oils, the one that will keep the bearing the coolest is the best lubricant.

The author emphasizes the necessity of oil-recovery in connection with continuous oiling systems, and gives instances of great waste where elaborate provision has been made for applying the oil, but none for its recovery and re-use. Another point brought out is the loss which occurs in burning oily waste instead of recovering the oil contained therein.

In closing the paper gives directions for the systematic distribution of oils to employees, and outlines a method for keeping accounts. An appendix gives the plans and specifications of the oil-house at the Baltic mine..

Thursday, August 29.

The itinerary for the second day included points of interest north of Houghton, notably the great stamp mills of the Calumet & Hecla Mining Company at Lake Linden. The general features of the milling process here are quite well known; nevertheless there has been enough progress of late to make the plants interesting even to those who had visited them before. The improved system of classifying and jigging which is now in operation is an innovation since the last meeting of the Institute at Houghton. Further, the project to regrind and reconcentrate much of the old accumulation of tailings struck the visitors as conclusive evidence of the improvement in modern milling methods.

At Ahmeek there was a good opportunity to see mass copper of various sizes, just as it is sent from the mine to the smelter. Several carloads of mass were awaiting shipment. One of these consisted of two large pieces of copper, weighing perhaps 15 tons each. Other cars were loaded with smaller masses. Another matter of interest at Ahmeek was a concrete "timber" set, which had been assembled in the position which it would occupy in the shaft. This system of shaft lining is used to the complete exclusion of timber, and presents some possibilities in extending the application of concrete underground. The first cost is less than for timber, and the resulting structure is fireproof and indestructible in ordinary service.

At noon the party enjoyed an hour of recreation at Electric Park, where lunch was served. Proceeding then to Franklin, the mining and mechanical engineers were much interested in watching the operation of the air-balanced hoisting engine which is used on the single-compartment shaft, hoisting ore in 10-ton skip-loads. During the descent of the skip the engine works as an air compressor, storing air in great receivers under an ultimate pressure of about 95 lb. per square inch. When starting to hoist, the compressed air is used as long as available, after which the steam automatically comes on. This arrangement relieves the steam plant of the peak load required to start the hoist and accelerate it to normal speed.

The social feature of the meeting was a dinner at the Onagaming Yacht Club, where the visitors were taken by boat. The Calumet & Hecla band provided music on the boat and rendered an excellent program after dinner.

Bureau of Mines.

The speaker of the evening was **Dr. Joseph A. Holmes,** director of the Bureau of Mines, Washington, D. C., who had come to Houghton at the special invitation of the Institute. Dr. Holmes spoke at some length on the function of the Bureau of Mines and outlined some of the relations which it would and would not have with the mining industry. The pubic welfare is the only basis for the Bureau's work, and unless the results of this work contribute to the public good the activity of the Bureau is not justified.

There are two phases of public welfare which Dr. Holmes thought worthy of the Bureau's attention and within the scope of its plan and purpose. These are public safety and avoidance of waste. On these subjects it is the desire of the Bureau to gather and disseminate all the information which it is possible to obtain, from foreign countries as well as our own. Private or secret information is not sought, neither is it the intention to encroach on the field of private endeavor, but, on the contrary, to maintain that strictly public position which shall make the Bureau useful.

Dr. Holmes stated that as a result of the Bureau's work mine disasters and accidents have been steadily diminishing, although they have not yet been reduced to the desired minimum. It has been the aim of the Bureau to instil into the leaders of the mining industry ideas relating to the safety and discipline of the miners, and to secure their co-operation in making mining less hazardous. Practical instruction is given in handling explosives, and mine rescue work is taught.

On the second phase of his subject Dr. Holmes reminded his audience that conservation of mineral resources did not mean that we should not use those resources to just as full an extent as our conditions demanded, but that it did mean that we should so conduct our operations that great loss of resources would not result. He cited the fact that by our present system of coal mining we waste half a ton for every ton mined; that more natural gas is wasted annually than is produced artificially. These are the greatest examples of mineral waste and need of conservation. An appreciation of the situation is already felt among the metal-mining companies, for a vast amount of low-grade iron ore which is now worthless is being saved in anticipation of the day when it will be needed and when improved metallurgical methods will make it avail-

able. Other ores of low grade throughout the West are being conserved and treated, in full realization of the fact that the bonanza days have gone by.

Following Dr. Holmes' address, Mr. M. E. Gow commended the work of the Bureau and advised all present to take advantage of the information contained in its free publications. Mr. Gow then spoke briefly of the welfare work of the Steel Corporation in behalf of its employees.

Friday, August 30.

The closing session of the Institute was held Friday morning at the School of Mines and was devoted to reading and discussion of papers and election of officers.

Recent Rock-House Practice in the Copper Country.

There were two papers presented on this subject; the first was by **Tenney C. DeSoller,** of Hancock, Mich., who described the practice at No. 2 rock-house of the Quincy Mining Company as representing the best procedure which the company has yet been able to devise.

The Lake Superior methods of handling copper rock, as it is locally termed, have undergone many improvements during the past few years, both with respect to underground and surface operations. Past experience has shown that it is advisable to break the rock to a size that can be advantageously fed into the steam stamps at the mill. The material, as it is hoisted to the surface, comes in sizes varying from fine material to large masses of native copper, copper rock and poor rock, weighing at the maximum several hundred pounds.

General Description of Building.

The building is of steel, with corrugated iron roof and sides, and reinforced concrete foundations and floors, thus making the building absolutely fireproof. All the bins are of steel and are circular in transverse section. The bottoms of the bins, instead of being built up and costly to maintain, are filled with poor rock until the latter assumes an angle of natural slope, thus making its own bed and causing all copper rock dumped into the bins to discharge through the regular openings. The building is 150 ft. long and 30 ft. wide, except where the large stamp rock bin, 44 ft. in diameter, is located. The crusher floor, situated on top of the stamp rock-bin, 45 ft. above the discharge aprons, is of the same size but square. This bin will hold upwards of 2000 tons. This large capacity takes care of variations in car supply and rate of hoisting.

The skip track above the collar of the shaft has an incline of 54 deg. and the shaft runners are carried up on this angle to the copper rock dump, where there is an outward curve toward the hanging of the shaft for the wider face of the rear skip wheel to travel upon. The 12-ft. wood-filled rope sheaves are located 119 ft. vertically above the collar of the shaft and are securely stayed by means of a batter brace. A room located at one side of the collar of the shaft is occupied by the lander, one of whose duties it is to transmit to the hoist engineer the signals as communicated to him from the underground chute men. The lander, without leaving his room, moves a sliding plate by means of levers which allow an opening for the narrow face of the front skip wheels to pass through onto a track toward the foot of the shaft, the rear skip wheels, which have a wider face, continuing in the plane of the incline and thus dumping the skip.

There are three points of discharging: The first for mass copper and dull drills; the second for poor rock from sinking shaft or bottom level cross-cuts; the third, the regular copper rock discharge. A set of levers, one for the first and one for the second, will permit the opening of the discharge as indicated by the signals. An 8 x 8-in. steam hoist, located in one corner of the building, is for handling timber, cranes and other supplies; also for the track forms used to take a damaged skip or other conveyor from the skip road. The man-cars, water skip and rock skips are suspended from cranes and can be quickly swung into place and put in operation.

Rock Handling.

The rock is hoisted in the two 8-ton skips running in balance. The skips, as before stated, have three discharges. The first discharge handles the large mass copper and dull rock drills. In this dump the skip discharges onto a concrete block faced with old rail (flange up) having a slope of 30 deg. This slope changes to level and forms a platform of the right height for loading the mass copper onto a railroad flat car. Masses weighing up to 8 tons are loaded upon the railroad cars by means of 8-ton chain blocks hung from trolleys in either compartment carried by 18-in., 55-lb. I-beams extending out over the railroad track. The dull drills are easily loaded onto a wagon or car and taken to the drill shop to be sharpened, while the mass copper is shipped directly to smelter.

When poor rock is hoisted from shaft sinking or the bottom level cross-cuts, the second dump is opened by means of the above-described levers and the skip discharges into a reinforced concrete chute, which empties into a cylindrical steel bin 13 ft. in diameter. This rock is drawn off by two chutes, one of which feeds a 24 x 18-in. Blake type rock crusher, where the rock is crushed to a size adapted for concrete or road work. Under the crusher is a cylindrical steel bin 9 ft. in diameter, where it can be loaded from a chute into railroad cars or wagons. The chute on the opposite side of the 13-ft. bin permits the discharge of the coarse rock directly into a vertical cylindrical steel tube 5 ft. in diameter, dropping upon a reinforced concrete block slightly inclined, which breaks the fall, and from which it can be loaded through a chute into railroad cars, or wagons.

When copper rock is hoisted, the third, or regular, discharge is used and was made as above described in order that the lip of the skip might remain in a position close to the steel dumping plate and not cause the rock to be violently thrown from the skip. It might be stated that the copper rock dump was designed in order that the skip should discharge into a pocket which would fill until the rock took its natural slope, the rock then discharging onto the grizzlies. It was found during the erection of the building that there was a mistake of about 3 ft. between the blue prints and the steel construction, and a steel dumping plate was substituted.

The dumping plate spreads the rock on 6-in. steel grizzly bars approximately 16 ft. long, set at 16 deg., and having 20-in. openings. Immediately above the grizzly bars is a battery of heavy bars, which serves the double purpose of breaking the fall of the rock and spreading the same upon the grizzlies. The oversize from the grizzlies passes down on a reinforced concrete chute, striking a second battery of bars, then drops vertically into a bin having its side open toward a 3000-lb. drop hammer. Between bin and hammer is a 15-in., 42-lb. I-beam carrying a traveling 8-in. 2-ton air lift. This I-beam is bent in the shape of a horseshoe and permits the air lift to be used at either crusher, or poor rock chute. Here the oversize, if mass copper, is cleaned of the poor rock under the hammer; if copper rock, it is broken to a size that can be handled by the crushers; or, if poor rock, it is thrown into a chute leading to the poor rock bin.

In case the broken oversize is small mass copper, it is thrown into a mass copper chute leading into a cylindrical steel bin

6 ft. 8 in. in diameter, which discharges into railroad cars. If it is mass copper too large for this chute, it is loaded onto a pan, swung from a crane, and lowered outside the building to a reinforced concrete platform at the right elevation for loading on a railroad flat car.

The undersize from the 20-in. grizzlies falls upon a second grizzly composed of 3½-in. steel bars, 6 ft. long, set at 30 deg., and having 2¾-in. openings. The undersize from these last grizzlies passes directly into the stamp rock bin below the crusher floor. The oversize is discharged into a cylindrical steel bin 14 ft. in diameter holding approximately 10 skip loads of rock. The outlet from this bin is by two chutes 19 ft. apart. Vertically sliding steel doors, 4 x 4½ ft., with replaceable steel linings, operated by 6 x 36-in. air cylinders, control the feed into steel chutes set at 30 deg. This feed is directed into the two 36 x 24-in. Blake type rock crushers and regulated by a hinged apron, which is operated by a 6 x 18-in. air cylinder. One man at each chute feeds the crusher, picks out the poor rock and mass. The poor rock goes into a chute leading to the poor rock bin. The mass is stored upon an inclined semi-circle chute called the "copper pan," on which it easily slides to a small steam hammer, is cleaned from poor rock, and
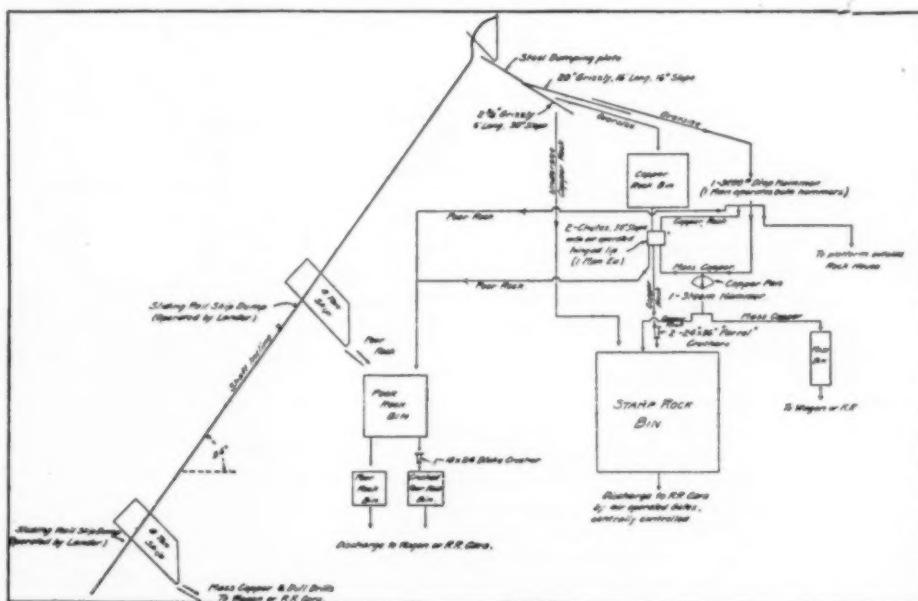

FIG. 1.—FLOW SHEET OF ROCK HOUSE NO. 2, QUINCY MINING COMPANY, HANCOCK, MICH.

thrown into a chute leading to the mass copper bin. This steam hammer is run by a third man who handles the trolley already described, cleans the mass copper, oils, and has charge of the machinery.

The crusher jaws are made of manganese steel, and are set to crush to 3 in. The crushers, running at 140 r.p.m., are operated by a 12 x 24-in. Nordberg Corliss-valve steam engine running at 110 r.p.m. A 75-hp electric motor occupying less space is installed, and is ready to run in case of a breakdown to the engine..

The rock from the stamp rock bin is loaded into railroad cars for the mill by means of discharge aprons operated by 5 x 13-in. air cylinders.

Three men handle upwards of 1000 tons of rock every 12-hour shift, change man-cars and skips on skip-road and load all timber and supplies that are lowered underground. Between shifts, when the large mine air compressors are not in operation, power for the operation of the air lifts and air controls is furnished by a small 11 x 11 x 12-in. Westinghouse air pump, so arranged that when the air pressure from the mine compressor drops, the air pump automatically starts. A com-

bination of check valves prevents loss of this air into the underground system and, likewise, when the air from the compressor is raised, the air pump is automatically stopped. This system of handling material has shown an average rock house cost of less than two cents per ton.

Rock-House Practice at Copper Range Properties.

The second paper was by **H. T. Mercer**, Painesdale, Mich., who described the practice of the Copper Range Consolidated Company. Mr. Mercer first reviewed the early practice, showing the development of the dumping system from the time when ore was hoisted in 2½-ton skips to the present time when 6-ton skips are used.

In our old rock houses, the rock on leaving the skip dropped on an iron door about 5 ft. long, set at an angle of about 30 deg., below which, set at the same angle, were the screen bars, or grizzlies. The iron door was hinged at the lower end and could be raised to allow for the dumping of waste rock into a separate bin or car, underneath the skip track.

The grizzlies were first constructed of 4-in. round iron bars, about 12 in. long, set 4 in. apart. These were later discarded in favor of 3 x 10-in. hardwood bars capped with 1 x 4-in. iron straps. The latter were cheaper and easier to replace than the round bars. At the lower end of the screen was a 4-ft. drop to the floor, and at this point in the floor several bars, or rails, were inserted, set 4 in. apart, at right angles to the grizzlies, forming a floor screen. Any fine dirt that had failed to drop through the grizzlies could be shoveled through this floor screen to the storage bins below.

The crushers used at that time were of the jaw pattern, 18 x 24-in. in size, and ran at a speed of 85 to 90 r.p.m. There were two of these crushers, one in front of each grizzly, and about 12 ft. away, the floor between the crusher and the grizzly being paved with heavy plates. This long space was to allow for sorting and picking of mass, but necessitated a great deal of shoveling to get the rock to the crushers. The mass copper was picked out and taken, by hand, across the floor to a drop hammer, where the surplus rock was pounded off. It was then thrown through a chute to the ground, where it was collected and teamed to a central point, to be loaded onto a car and shipped to the smelter. Pieces of rock too large for the crusher were also taken to the hammer and broken. The finer pieces resulting were dropped through an opening in the floor to the bins and the large pieces taken back to the crushers. Beneath the crushers were the large storage bins, from which the rock was loaded into cars by means of the ordinary apron chutes operated by separate hand levers. Any waste sorted on the crusher floor was transferred by hand or barrow to the poor rock car, which ran on a track a little below the level of the main floor. Most of the waste, however, was hoisted separately, dumped directly into the poor rock car and trammed out on to the waste rock pile by the rock house crew.

Such, briefly, were the methods in use ten or twelve years ago. Let us turn now to the newer method and see what has been accomplished. In the following description, reference is made to Fig. 2:

In the first place we have done away entirely with the grizzly. On leaving the skip the rock drops directly into a large bin, capable of holding about 20 or 25 skip loads. In the front side of this bin there are two sliding doors, one in front of each dump, each operated by an 8-in. air lift. These doors are 5 ft. wide and 5 ft. high and are lined with 2-in. cast-iron plates. The front of the bins above the door is also lined with iron to protect the wall from the impact of the rock. The bottom of the bin is about 2 ft. below the bottom of the doors and this space is filled with waste rock until it takes its natural slope

to the door. No paving is used except near the door, where a heavy iron plate projects into the bin at an angle of 30 deg., to help in starting the rock through the door. Outside of the bin in front of each door is an apron or chute about 4 ft. long, set at 30 deg. The lower end of the chute rests directly on the jaw of the crushers, which are Farrell type 24 x 36 in. in size, run at a speed of 155 to 185 r.p.m. They are designed for as high as 300 r.p.m. There are two of these crushers, one in front of each door. Each crusher has a capacity of from 500 to 1000 tons daily. Fastened to a shaft on the top of the crusher, by means of arms, is a cradle or gate, which drops into the lower end of the chute. This gate is operated by a 4½-in. air lift. The operation of feeding the rock to the crusher is as follows:
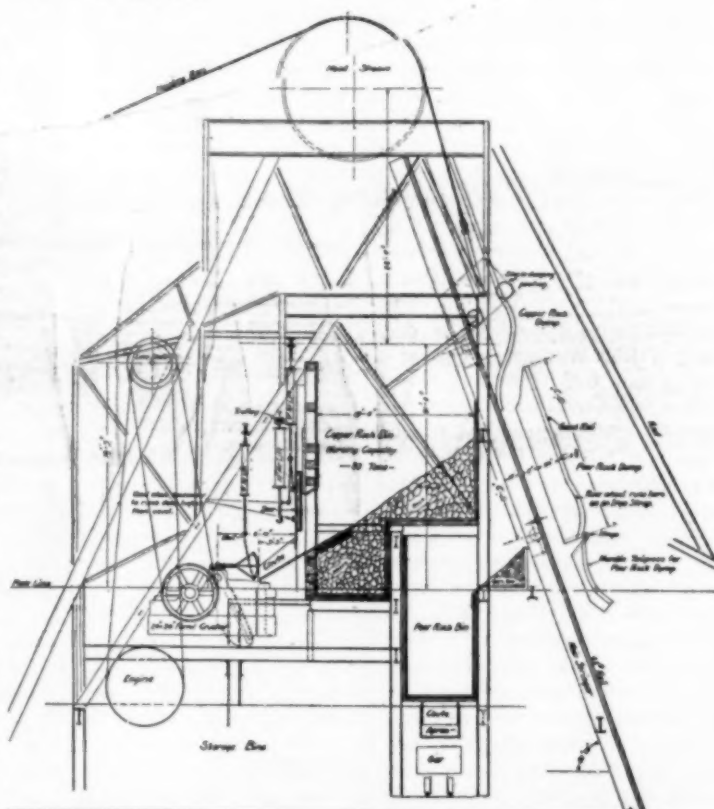

FIG. 2.—SECTIONAL ELEVATION OF ROCK HOUSE OF THE CHAMPION COPPER COMPANY.

When the bin door is raised the rock runs down into the chute and is stopped at the crusher jaw by the cradle gate. The bin door is then shut and any mass or large boulders that may have come down are sorted out. Then the gate is raised and the rock feeds directly into the crusher with little or no handling. A trolley, carrying a 10-in. air-lift, runs directly over the chutes in front of the crushers, and by this means the mass and large rocks are quickly picked up and transported to the drop hammer, which is set at one end of the building, and operated by a small electric hoist. On leaving the hammer the mass is pushed through a door in the side of the building and drops straight down to a platform alongside of the railroad track. This platform is built of log cribbing filled with waste rock, the top being at the same elevation above the track as an ordinary flat car. Hinged on the side of the rock house above the car platform, in such a position that it swings out over the car, is a simple jib crane, built of an 8-in. I-beam, 10 ft. long. On this beam runs a trolley to which a chain block can be attached. By this means the mass copper is easily loaded onto the car. The larger masses are picked up with a chain, and an iron pan swinging on chains, is used for the smaller pieces. No sorting of poor rock is done in the rock house. All waste

hoisted is dumped into a separate bin by means of a lower hinged dump such as has already been described. From this bin it is loaded into a car trammed out on a trestle and dumped through a raise into the old stopes.

At several of our shafts where the poor rock trestle is of sufficient height bins have been built near the end of the trestle, about 6 ft. back from the hole, or raise. Over these bins just under the car track, is a screen or grizzly, about 10 ft. long, with bars 1½ in. apart, set at an angle of about 35 deg. When the car is dumped, the waste rock slides over the grizzly. The larger pieces go over into the raise and the fine drops through the screen into the bin, where they become available for road material and concrete. When no rock is being drawn from the bin it simply fills up to the screen, and everything goes over into the mine. At one of our rock houses, where the old screen bars are still in use, and the crushers are set 12 ft. away from the screens, large feed pans or "tilting" pans have been installed, similar to those in use at some of the Calumet & Hecla rock houses. The pans are hinged at the crusher end, and extend from the crusher to the grizzly. At the screen end they are attached by means of a bail and chains to a large air lift, by which they are operated. On being raised they stand at an angle of about 38 deg. and feed the rock to the crushers.

The main storage bins at the older rock houses are of the rectangular pattern, with sloping bottoms. The rock is drawn from these bins by means of apron chutes, operated by long levers. Of late years most of the rock houses built are provided with large circular steel tank bins of great capacity. The loading chutes in these circular bins are set in the bottom directly over the center of the railroad track. They are an improved pattern closed by small doors, operated by a system of levers from one central point. These new style chutes are more easily operated than the old aprons, and will probably replace the latter at our older rock houses.

Let us now consider what has been gained by these changes: First, the large dump bin obviates the necessity of stopping the hoist in case of an ordinary delay at the crusher, such as the stopping of the crusher by a mass, etc. Second, we do away with screen bars and the expense of their up-keep. Under the old system Sunday repair work was a big item; since introducing the direct feed scheme, Sunday repair work has been almost entirely done away with. The sliding doors and chutes last from two and one-half to three years or longer, as do also the cradle gates at the crusher.

Perhaps the biggest saving, however, is in labor. Formerly a rock house crew consisted of six to eight men. At present two men will handle easily all that is hoisted.

Handling of mass copper and large boulders has been made comparatively easy by means of air lifts, trolleys and cranes, and whereas in olden times a rock house job was about the hardest around the mine, it has now become one of the easiest. Finally, to sum up the advantages in terms of actual saving in cost the comparison of rock house expense before and since remodelling is as follows:

Year.	Tons Crushed.	Labor.	Cost per ton Supplies, etc.	Total.
1906	671,785	$0.0629	$0.0342	$0.0971
1911	734,392	0.0262	0.0283	0.0545

Balancing Rock Crushers was the title of an interesting paper by **Mr. O. P. Hood**, of Pittsburgh. It is well known that heavy machines in which there is a vibratory motion, require specially rigid foundations in order that harmful vibration shall not be transmitted to the structure. Sometimes when several crushers are placed near each other the cumulative effect of their vibrations when the latter are in phase is very noticeable and is sometimes difficult to overcome.

The idea of balancing the moving parts of such machines so that the forces tending toward vibration would be counteracted, was the basis of Mr. Hood's paper. Some encouraging results have been obtained up to date, and the investigation is being continued. It was Mr. Hood's opinion that the prin-

ciple of balancing such machines could be applied with resulting economy of power and efficiency of operation.

The following papers completed the program: "Some Applications of Concrete Underground," by H. T. Mercer, of Painesdale, Mich.; "Construction of Intakes at the Champion and Tri-Mountain Mills," by Edward Koepel, of Beacon Hill, Mich.; "The New Franklin Hoist," by R. H. Corbett; "Methods of Mining Iron Ore," by F. W. Sperr, of Houghton; "Development of Lake Superior Copper Mines," by F. W. Sperr; and some theoretical geological considerations, by Dr. L. L. Hubbard, of Houghton.

Election of Officers.

At the business meeting the secretary reported an enrollment of 486 members, and a cash balance in the treasury of $5,994. During the current session seventy applications for membership were received. The following officers were elected for 1913.

President—Pentecost Mitchell, Duluth, Minn.

Vice-presidents—F. J. West, Duluth, Minn., A. D. Edwards, Atlantic, Mich., W. P. Chinn, McKinley, Minn., to fill vacancy.

Managers—G. S. Barber, Bessemer, Mich., W. H. Johnston, Ishpeming, Mich., C. H. Baxter, Loretto, Mich.

Secretary—A. J. Yungbluth, Ishpeming, Mich.

Treasurer—E. M. Hopkins, Commonwealth, Wis.

The last act of the Institute was to pass resolutions on the death of Graham Pope, who was early associated with the Institute, and one of its ex-presidents.

The place of meeting next year has not been decided upon, but it probably will be in the iron country of Minnesota, in conformity to the usual practice.

Michigan College of Mines.

After the close of the Institute, the College of Mines acted as host to the visitors and threw open the laboratories for inspection. Small squads were guided by students to the different places of interest. Special demonstrations had been arranged in timber testing, petrography, wireless telegraphy and metallurgy.

Spelter is being produced this year at a greatly increased rate as compared with 1911. C. E. Siebenthal has compiled the figures for the first six months of 1912, and finds that if the same rate of production is continued during the last half of the year, the output of spelter will amount to about 330,000 tons, or 46,000 tons more than in 1911. Kansas, Illinois and Oklahoma are the principal spelter-producing states, ranking as named. The total capacity of zinc smelters reporting production at the beginning of this year was 81,582 retorts, to which it was contemplated to add 15,868 retorts in new construction. Allowing an annual average capacity of 4¼ tons per retort, the capacity of the 81,582 retorts for the first six months of 1912 would be 173,362 tons, less than 7000 tons in excess of actual production.

The Utah Copper Company's report for the second quarter of 1912 contains the following interesting data. The gross production of copper in concentrates was 9,457,346 lb., recovered from the treatment of 1,503,884 tons of ore containing 1.42 per cent copper. The average net cost of copper was 8.127 cents per lb. The Magna plant treated about 70 per cent and the Arthur plant 30 per cent of the tonnage. The latter has been remodeled, and its thirteen sections are ready for operation at this writing. Some improvements still remain to be made in the coarse crushing department, in order that full capacity may be maintained during the winter months, when the ores are more difficult to handle.

Cripple Creek supplied 56 per cent. of the total gold yield of Colorado in 1911, with an output of $10,562,653. Conditions in this district were much the same in 1911 as in 1910, for the subsidence of the water through the Roosevelt tunnel was so slow that it was not possible to materially increase operations.

Electrical Resistance of Sulphates.

BY A. A. SOMERVILLE.

Copper sulphate, $CuSO_4$, and iron sulphate, Fe_2SO_4, were the two compounds experimented upon.

Copper sulphate is the ordinary blue vitriol of commerce, commonly used in making what is known as the gravity cell for telegraphic use. When the crystals are heated so as to drive off the water they turn white. The iron sulphate crystals
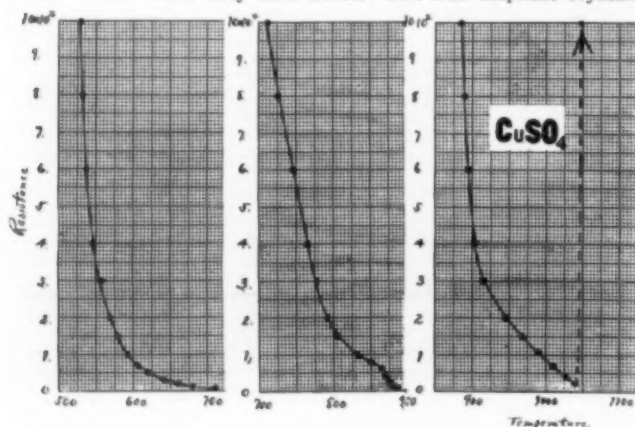

FIG. 1.—CHANGE OF RESISTANCE OF POWDERED $CuSO_4$ WITH
TEMPERATURE.

are white but tinged with a yellowish brown color. When heated they do not change color until a new compound is formed.

In each case the crystals were pulverized and the powder placed in a porcelain tube that could be heated. Terminals were placed in the ends of the tube so that the electrical resistance of the sulphate could be measured. The heating was done in an electrical furnace and the resistance measured with a resistance thermometer connected to an automatic recorder.
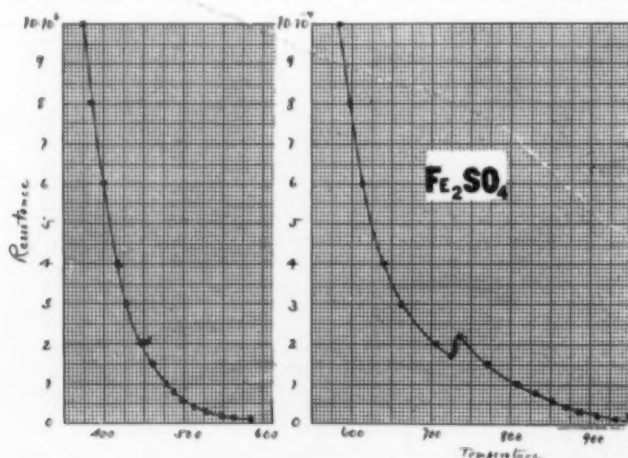

FIG. 2.—CHANGE OF RESISTANCE OF POWDERED Fe_2SO_4 WITH
TEMPERATURE.

At about 865 degrees Centigrade, the copper sulphate begins to slowly and gradually decompose, forming $CuO + SO_3$, black copper oxide and sulphur trioxide. The sulphur trioxide is driven off at this temperature, leaving the black copper oxide in small quantity.

The electrical resistance of the copper sulphate is infinite at room temperature. The powder was contained in a tube one centimeter in diameter and three centimeters long. It measured 10,000,000 ohms at about 525 degrees. The resistance decreases in a regular manner until about 860 degrees is reached. Here the rate of change of resistance is altered due to the decomposition beginning. As the sulphur trioxide is driven off, leaving

a smaller volume of copper oxide than before, the resistance decreases relatively faster. At about 1030 degrees the cross-section of material left is so small that the electrical circuit was broken as indicated by the apparent infinite resistance again.

The iron sulphate decomposed quickly and suddenly at about 730 degrees Centigrade, forming $Fe_2O_3 + SO_3$, iron oxide and sulphur trioxide, drawing oxygen from the air to complete the transformation.

The iron sulphate measures 10,000,000 ohms at about 375 degrees. Its resistance decreases similarly to that of the copper sulphate. At about 730 degrees there is a sudden decomposition and the sulphur trioxide is all driven off in a short time, making an abrupt break in the temperature-resistance curve. From there on the resistance is the same as for iron oxide.

The points where decomposition begins are very definitely shown on the temperature-resistance curves.

An Electric Still.

In the accompanying illustration, Fig. 1, is shown an electric still adapted for difficult distillations, which was described in our Congress issue, page 606, but not illustrated there. The still was devised by **Mr. Irving C. Allen,** of the U. S. Bureau of Mines, Pittsburgh, who described it before the section on organic chemistry during the sessions of the Congress.

Turn from a block of wood a model the size and form of the distilling flask, with the bulb and neck in one piece, and saw the model into halves from top to bottom. Hollow out the bulb and neck to form a shallow dipper making the cavity about 1 cm. deep, and leaving the rim about 3 mm. thick. Drill holes the size of a small wire brad 1 cm. apart and 3 mm. below the edge of the rim of the dipper and along the handle or neck as far as it is desired to have the heating wire. Insert brads through these holes from the inside and allow the points to project about 3 mm. thus forming a comb of brad points around the outside of the dipper near the edge, Fig. 1, a.

Then, after greasing well the convex surface of the mold to prevent the paste from sticking, wrap the heating wire back and forth, the wires being 1 cm. distant from one another, across
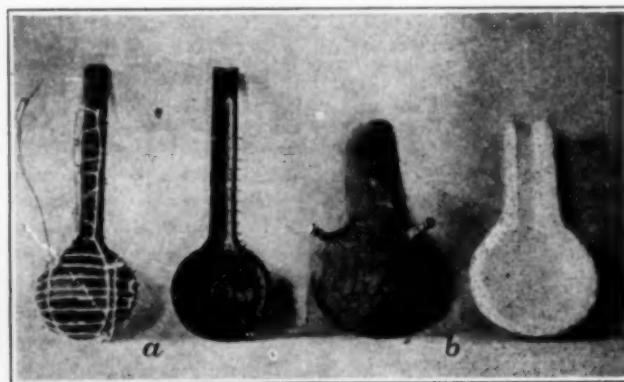

FIG. 1.—ELECTRIC HEATER AND MOLD.

the convex surface of the dipper between these points, in the manner shown in Fig. 1, a, tying the wire in place with asbestos cord and making its two ends fast to binding posts projecting at convenient positions. Lay the wired form, convex side up, on a smooth board covered with a sheet of paper and apply freshly prepared paste to the desired thickness, about 2 cm.

Allow the paste to harden over night. Remove the brads from the inside of the mold. The heater will then be released and ready for the final drying described above. Do not pass a current through the wire until the heater has been thoroughly dried, else rapid corrosion of the wire will result.

Final Meetings of the Eighth International Congress of Applied Chemistry.

In our extra issue of September 12 a full report of the proceedings of the Congress up to the night of Wednesday, September 11th, was given. After this date there were only the final sessions of the Sections of Thursday, September 12, the banquet on the evening of the same day, and the closing business meeting on Friday, September 13.

The banquet at the Waldorf-Astoria, on Thursday evening, was the great social function of the Congress.

The grand banquet was attended by over 1000 men and women members of the congress and was the occasion of many expressions of felicitation by visitors from foreign lands. On behalf of our own country there were addresses by representative public men.

Dr. William H. Nichols, president of the congress, presided at the banquet. In the absence of Governor Dix the State of New York was represented by Dr. John H. Finley, president of the College of the City of New York, who spoke to the evident delight of his audience.

Mayor Gaynor's Address at the Banquet

The principal speaker of the evening was Mayor Gaynor, of New York City, who was the guest of the chemists. In introducing Mayor Gaynor Dr. Nichols spoke of him as "the man who has the hardest job in the world." Below we give the text of the Mayor's address:

"As I heard the gentleman call off the announcements in five languages, I could not help wondering all the while whether there were not possibly seven or eight languages in which he could keep silence. Or maybe he could keep silence in all of them, which is better yet. I knew my job was a difficult one without the chairman announcing that to this audience. I think it is pretty well known, and I also know possibly that the harder you work in it to do right the less thanks and the more abuse you get from some quarters here in the City of New York. But the quarters, after all, are quarters that do not count for much.

"I desire to make an apology to you. If this Congress had been held in any of the cities throughout the country from 20,000 and upward the newspapers of those cities of 20,000 and upward would have devoted more space to your proceedings than the newspapers of this city have done. I cannot refrain as Mayor of the city from making to you an apology on that score, for the like of it I do not think you have ever seen before, and I am sure you will not encounter it in St. Petersburg next year. Still it is very natural. The papers here of large circulation, who make daily affidavits, each of them, that its circulation is the largest in the whole world, dole out their ideas in types from an inch to 6 in. long. And not being content with black type even of that length, some of them, as you must have noticed, Sir William, and the rest of you, have to use red ink.

"They think our skulls are so thick that there is no other way of penetrating them, even with the commonplace thoughts which they seek to convey. But I may say to you that these papers speak principally to the underworld of this town, the great underworld of 200,000 in this great city of 5,000,000, and that underworld are not scientists, I assure you. And it needs big type, and red type, to penetrate their skulls, and some of them have skulls so thick that possibly you could not get anything into their heads, except in the—I don't know whether I ought to say this—except in the same way that you get a joke into a Scotchman's head—that is to say, by taking out a section of his skull and poking it in.

"You men are the forerunners of human thought and of progress throughout the world. You make the world move forward. You are in the very front. You have revolutionized things throughout the world, and yet, may I say to you, that with all you have done we are still only on the edge of things. Isn't that so? We are only beginning. And yet you are working, experimenting, bringing out new processes and ways of applying them. The manufacturers of the world are now making fortunes out of things which a few years ago they threw into the waste heap, all through the science which you have developed. I hope you get your share of it.

"And yet, as I sat and looked at you to-night and saw how well and prosperous and well paid you all looked, I said to myself, what condition will they all be in when the impending state of socialism comes into reign, and a great many people think it is now to come to pass very soon. The Socialists have not yet determined whether they shall pay every man according to his necessities, every worker according to his necessities, or according to his productive capacity, whether mental or physical, or both. If they determine to pay only according to your necessities, that is to say, to each just enough to make him live, you people will have to come down a peg or two under socialism. And I am afraid you won't work so mighty hard any more. I am afraid the development in chemistry will come to quite a standstill when that comes to pass; and that will be true in all the sciences, and in all departments of human endeavor, and in all the arts. When everybody is paid alike I think we will all try to live alike and do as little as we can."

Mrs. Baekeland on the Advancement of Women.

The unique place of honor among the list of speakers was awarded to Mrs. Leo H. Baekeland, who was the only woman to address the Congress. Mrs. Baekeland acted as chairman of the Congress ladies' committee and contributed greatly to the success of the meetings in Washington and New York by arranging for the entertainment of the ladies.

In her brilliant and dignified address at the banquet she took for her subject "The Advancement of Women," speaking as follows:

"Mr. President, Ladies and Gentlemen:

"I am not a real chemist. I am only a chemist by birth, by marriage, by friendship; and so, it is as a woman that I will speak to you to-night, as a woman and as a friend.

"It has often been my good fortune to be amongst you; and I have derived so much strength from the atmosphere of intelligence and broad-mindedness which prevails at all your meetings, that I think it is greatly due to this influence that I have become what is called a new woman.

"Some time ago this would have been a confession; to-day it is a declaration of faith. Some time ago the new woman was the butt of all the stale jokes; to-day she is the deep concern of all thinking men.

"Do not imagine that all the new women want to break windows; I know some very charming ones who are only breaking hearts. I know some very fine ones who are trying to mend hearts.

"If man or woman, if any human being could have the mental ability of man—and at the same time the charm and tenderness and love of woman, this being would at once take a great step towards human perfection. Such a step is not at all impossible now, and such women should exist and do exist—women of the type of Jane Addams, who works with the directness and power of man and with the devotion and love of woman.

"Do not be frightened by the cry of equality, by the cry of

rights for women; it is the cry of a dignified being, asking for a dignified place in society. Not the place of a toy, not the place of a pet, not the place of a spoiled and capricious child as so many of us unfortunately are, but the place of a dignified being.

"We know that we have much to learn, but we are willing and ready to do so; and when we see a gathering like this one, where hundreds of ladies have come from all parts of the world to join a Congress of Chemists, we know that 600 ladies have had a chance to improve and to grow.

"It is to your generosity, gentlemen, to your cordiality, that we owe these few delightful days of education, of good fellowship, of advancement and in the names of all the ladies present I thank you."

Addresses by Delegates.

A large number of representatives of foreign countries spoke briefly on the value of the international congress and the success of this meeting. The following list of speakers shows the world-wide nature of the gathering.

William Percy Wilkinson, representing Australia; Rudolph Wegscheider, of Austria; B. Setlik, of Bohemia; Paul Hagamans, Consul General of Belgium at Philadelphia; Anthony McGill, of Canada; Belisario Diaz-Ossa, of Chili; Yung Kwai, of China; Don Guillermo Wiles, of Colombia; Einar Biilmann, of Denmark; Leon Lindet, of France; von Buchka, of Germany; Rudolph Hofmann, of Prussia; Gilbert T. Morgan, of England; Giuseppi Bruni, of Italy; Adolfo P. Castanares, of Mexico; Alfred Sinding-Larson, of Norway; Mirza Ali Kuli Kahn, of Persia; Paul Walden, of Russia; Knut Wilhelm Palmaer, of Sweden; E. P. Treadwell, of Switzerland; James Moir, of the Union of South Africa; Edward Robinson, director of the Metropolitan Museum of Arts and Sciences; Edmund Otis Hovey, American Museum of Natural History of New York; President Arthur Little, of the American Chemical Society; Dr. Strohmer, of the International Commission of the International Congresses of Applied Chemistry, and Sir William Ramsey, of the International Commission of International Congresses of Applied Chemistry.

Ninth Congress to Be Held in St. Petersburg.

The last acts of the eighth congress were to determine the place of holding the next meeting and to take action on the resolutions which had passed the several sections. The ninth congress will be held in St. Petersburg in 1915. The president-elect is DR. PAUL F. WALDEN, professor in the Riga Polytechnic High School. The honorary president of the ninth congress is PROF. DEMETRIUS P. KONOWALOFF, of the University of St. Petersburg.

This was decided in the final business meeting held on Friday morning in the Great Hall of the College of the City of New York. This final meeting was also devoted to the consideration of resolutions.

Action on Resolutions.

The proposal to adopt Esperanto as one of the official languages for the ninth congress met with opposition on the part of the International Commission, and was rejected on the ground that it would be more appropriate for the next congress to pass on the matter as it could then be considered by a more representative cosmopolitan body.

An atomic weight table for commercial purposes, to be issued every five years and considered as a standard, had been proposed by the section of Analytical Chemistry. This was modified by substituting the official table of atomic weights now issued by the International Commission, adopting the table of 1912 as a commercial standard until the next congress meets in 1915.

A recommendation was adopted that at this congress and future congresses the reports of the International Commission of Analyses and of the other commissions appointed by the congress be printed in the transactions and an appropriation of $200 was made for this work.

A resolution directing the congress to investigate tests for hydrometers and other instruments, and to prepare tables on the density and expansion of solutions and fluids, was rejected.

Official notice was taken of the work of the committee on the stability of explosives, and it was recommended that a permanent committee of European experts be appointed to bring the committee's regulations to the attention of their respective governments and urge their adoption. The report of the committee will be printed in full and issued to the members of the congress.

Action was taken toward the appointment of a committee to confer with the International Hygiene Congress, to determine simple bacterial tests to control disinfectants.

The International Committee on Analytical Methods for Food Products was continued and an appropriation of $200 was allowed for its work.

The sub-committee on organic coloring matter in foods was continued, with instructions to suggest to all governments that they adopt identical regulations with regard to the use of these substances, and to endeavor to secure the co-operation of manufacturers in preparing a report on the reactions and analysis of organic coloring matters.

The congress took negative action on a proposal "to examine and report upon the progress and position of chemical industry in each of the countries represented at the preceding congresses, having particular regard to the relation between the development in the different branches of chemical industry and custom tariffs."

The work of the International Commission on Annual Tables of Constants received the moral support of the congress to the end that financial support might be obtained from scientific societies and institutions.

The committee appointed to prepare resolutions of thanks to societies, institutions and individuals for their support of the eighth congress presented its report, which was adopted. Following this, Dr. Carl Duisberg spoke appreciatively of the work of the president of the congress, Dr. W. H. Nichols, and asked that a vote of thanks be expressed to him for his services. This was done by a rising vote.

In conclusion, representatives of Belgium and Japan, who had not had an opportunity to speak at the banquet, made short addresses. President Nichols then declared the congress closed.

Additional Abstracts of Congress Papers

Section on Electrochemistry.

Ductile Tungsten.—In the last few lines of Dr. **C. G. Fink's** paper on page 581 of our issue of September 12, the table of properties contains a number of slight typographical errors. The corrected values are:

Young's Modulus of Elasticity: 42,200 kg. per sq. mm. (steel 20,000).

Melting Point: 3177 deg. (Langmuir); 3100 deg. $\pm$ 60 deg. (Pirani & Meyer).

Magnetic Susceptibility: $+0.33 \times 10^{-6}$ (Honda) i.e., practically non-magnetic.

Insoluble in HCl, H_2SO_4, HNO_3, HF, NaOH, KOH (aq), and mixtures of $K_2Cr_2O_7 + H_2SO_4$, soluble in mixtures of HF and HNO_3 and in fused nitrates and peroxides.

Electro-Analysis with Acidified Chloride Electrolytes.—In a paper on the electro-analysis of copper, antimony, bismuth and tin with acidified chloride electrolytes, by **Eugene P. Schoch** and **Denton J. Brown**, of the University of Texas, Austin, Tex., it is pointed out that solutions of chlorides of metals acidified with hydrochloric acid heretofore have been considered to be unsuitable for the electro-analytical deposition of metals, probably on account of the oxidizing effect of the liberated chlorine.

Since the liberation of chlorine can be prevented by the ad-

dition of a reducing agent, and since such electrolytes present all the advantages of simple acid electrolytes desirable for electro-analytical work, while in many respects they are more serviceable or convenient than either the "nitric acid" or the "sulphuric acid" electrolytes advocated by Foerster and by Sand, the authors decided to investigate their use, particularly with metals such as copper, antimony, bismuth and tin for which hydrochloric acid electrolytes are generally considered to be unsuitable and for two of which—namely, antimony and tin—the use of any simple acid electrolyte cannot be said to have been made practicable even by the recent excellent work of Sand.

The authors found that copper, antimony, bismuth, and tin may be deposited quantitatively and in good form from "hydrochloric acid" electrolytes if suitable reducing agents (formalin, hydroxylamine hydrochloride, oxalic acid, etc.) are added to them. By these means the authors have also found it possible to separate copper from tin and determine both from the same sample of electrolyte, thus reducing the otherwise troublesome analysis of bronze to a very simple operation; and they have found that metals with a potential "more zincic" than that of tin do not interfere with the determination of copper by this method.

The essential fact that makes this electrolyte serviceable is the presence of the reducing agent. Very probably the deposition of the metal takes place by the discharge of the "lower valence" ions, and the reducing agent serves, in effect, to change the "higher valence" ions to the "lower valence" ions.

These methods present many advantages over older methods. The electrolytes are easily prepared, because all of the metals are soluble in aqua regia, and all of their compounds are soluble either in this reagent or in hydrochloric acid. These electrolytes are free from odor as compared with the sulphide electrolytes frequently used for antimony and tin. The results indicate that the deposits obtained are probably free from inclusions; and the time required for the determinations characterizes these methods as "rapid" methods.

Details are given of the electrolytic apparatus employed, the determination of tin, the determination of copper, and its separation from other metals, the determination of antimony, and the determination of bismuth.

Inorganic Addition Agents for Deposition of Copper.—In a paper on the function of inorganic addition agents in the electrolytic deposition of copper by **Elwood B. Spear**, of the Massachusetts Institute of Technology, Boston, Mass., the theory is advanced that the function of inorganic addition agents in the electrolysis of copper solutions is to keep the copper in solution. The fundamental assumptions are: That some particles of copper may assume the colloidal form at the moment of giving up their electric charges at the cathode. That oxidation may take place on the cathode during the passage of the current.

Experimental proof is given to show that copper is continuously dissolved and reprecipitated at the cathode during the electrolysis of copper solutions.

The theory explains the fact that good deposits of copper become bad if the electrolysis is continued too long.

Carbon in Electrolytic Copper.—A paper by **Ellwood B. Spear, C. Chow** and **A. L. Chesley** shows that it is possible to precipitate carbon at the cathode during the electrolysis of a copper sulphate solution containing a little nitric acid and a small amount of gelatin.

Evidence is given to indicate that at least some of the carbon is in a free state in the deposit.

It is concluded from the experiments that an oxidizing action must have taken place at the cathode. This oxidation is probably due to secondary reactions.

Dry Cells.—A paper by **Carl Hambuechen** and **O. E. Ruhoff**, of Madison, Wis., gives some physical and chemical data on several typical American brands of dry cells, and on the materials used in their manufacture.

The chemical tests upon materials are made both for useless but inert impurities, and for actively injurious ones.

An important physical characteristic of raw materials used in battery manufacture, and of battery mixtures, is the electrical conductivity. The method of determining this is described and data are given upon various materials and mixtures. Conductivities depend upon the chemical composition of the materials or mixtures and upon the size and composition.

Tables are given showing dimensions and weights for several typical brands of cells which were investigated, and the chemical composition as regards the principal ingredients, namely, carbon and graphite, manganese dioxide, zinc chloride, ammonium chloride and water.

Carborundum.—In a paper on the study of a small carborundum furnace by **Wilder D. Bancroft, L. V. Walker**, and **C. F. Miller** of Cornell University, the authors give the results of tests in which they have studied the yields in a 30-kw and a 20-kw experimental furnace, as affected by length of run. A granular carbon core was used for the series of experiments. With the 30-kw furnace tests were made lasting 2, 2.5, 3, 3.5, 4, 4.5, 6 and 8 hours respectively. With the 20-kw furnace tests were made lasting 2.5, 3.5, 6, 10 and 14 hours respectively. Curves are given for core temperatures during a run, and for temperatures in one other part of the furnace. Curves are also given for yields of graphite, carborundum and siloxicon for different lengths of run. The results for the 20-kw furnace are given in Fig. 1.

The formation of graphite appears to follow approximately a straight line curve for some time, and then starts to bend at about the same time that the carborundum formation has reached its limit. The siloxicon curves show a slight bend in the opposite direction at this same point. It is almost self-evident that this would be the case, owing to the fact that the carborundum layer has arrived at a maximum radiation sur-
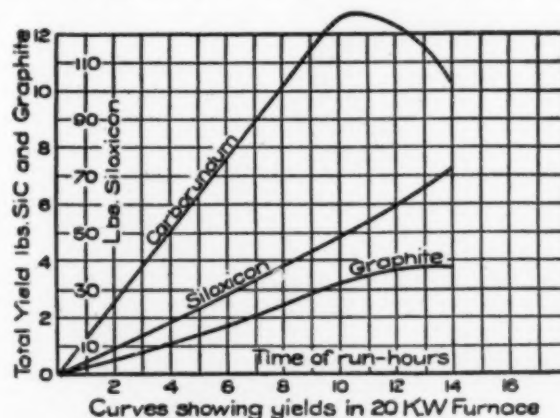

FIG. 1.—YIELD CURVES OF CARBORUNDUM FURNACE.

face as well as at a maximum temperature and hence it is most efficient in heating the siloxicon and unchanged charge about it. The slight decrease in the rate of formation of graphite is hard to explain except by the stationary stage theory, as there is always a layer of carborundum which could furnish carbon on decomposing, also an excess of carbon in the core itself. It may be that the carbon in the core is not as available owing to the size of the granules.

The carborundum curve shows a flat maximum, indicating that at this point the heat given off by the SiC is equal to the calories received from the graphite and core, none of the energy being expended to raise the temperature of the SiC layer. This is the beginning of the stationary stage. In the 30-kw furnace this maximum point is reached in six hours of run; in the 20-kw furnace at the end of 10 hours. The 14-hour run on the 20-kw furnace shows that the conditions for a stationary stage are by no means permanent in the carborundum furnace, and that there is such a thing as a rapid decomposition of SiC if

the run be continued too long. This could be explained as due to the heat-insulating effect of the then rapidly increasing siloxicon yield which would confine more heat in the SiC zone, causing the SiC to decompose at an increased rate.

Large Electric Furnaces.—A paper in German by **Dr. Rudolf Taussig**, of Vienna, Austria, gives a review of the "present status of the development of large electric furnaces" with especial reference to the Helfenstein patents, which have been repeatedly noticed in this journal. The furnaces are particularly intended for the manufacture of calcium carbide and ferrosilicon. One 10,000-hp Helfenstein furnace has been in operation for several months in France and is working satisfactorily. Other large furnaces are in course of erection.

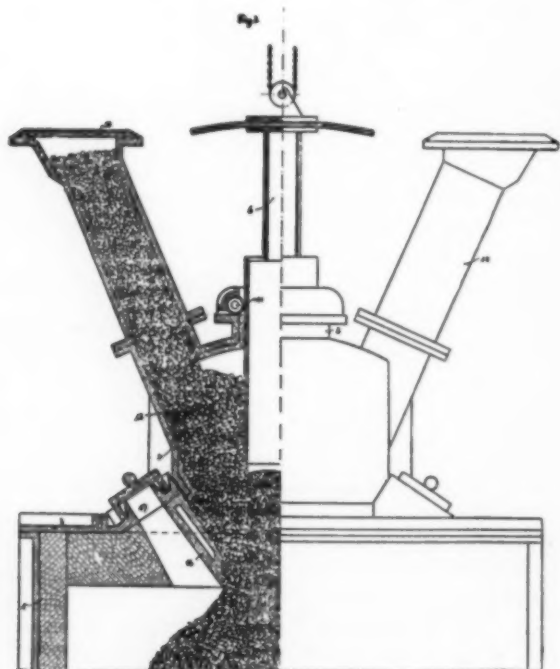In increasing the size of the electric furnace the difficulty

FIG. 2.—TOP OF LARGE ELECTRIC FURNACE.

was experienced that the arches of the roof would not stand the high temperature for a great length of time. This difficulty can be overcome by constructing a furnace of several shafts side by side, each with its own vertical top electrode, but all of them being connected together electrically by bottom electrodes in form of a conducting bottom common to all the shafts. Besides the electric connection there is no other connection between the shafts so that it is possible to produce different products simultaneously in the different shafts.

A second important improvement was made in closing the furnace airtight. The difficulty was in the method of feeding the charge and the way in which the upper electrode is suspended. Fig. 2 shows how this problem is solved by Helfenstein. Fig. 3 shows a 12,000-hp ferrosilicon furnace.

The increase in size has reduced considerably the cost of manufacturing. While with the smaller furnaces the price of calcium carbide was 18 to 20 kr. (1 krone = 20 cents), this was reduced by increasing the size of the furnace to 15 or 16 kr. and a further reduction to 12 or 13 kr. was obtained by using an airtight furnace.

The cost of making 50 per cent ferrosilicon was reduced by 20 kr. per ton by closing the furnaces airtight

The large closed furnaces are economical on account of their simplicity of the operation, the saving of electrode material and reduction in wages, and the smaller investment per ton of output. Also on account of the much lower maintenance cost. These large furnaces are operating for years without

reserve, since only the electrodes and the roof are used up. The replacing of the electrodes is very simple and the repairs of the roof can be made during operation.

While Helfenstein's furnaces were formerly used chiefly for calcium carbide and ferrosilicon, he is paying attention now also to other electric furnace problems, for instance, for reduction of iron ores and steel refining. The Stora Koppar-
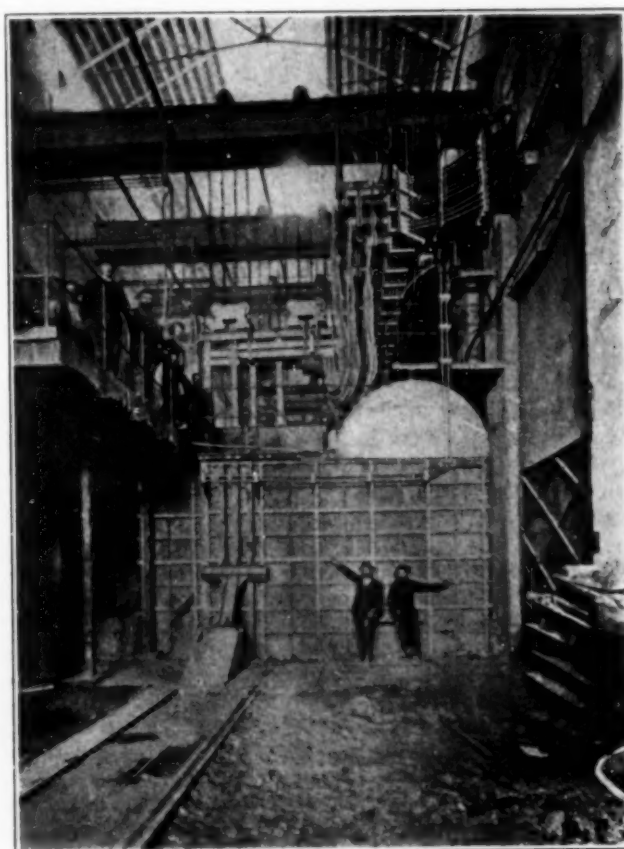

FIG. 3.—12,000-HP FERROSILICON FURNACE.

bergs Aktiebolag in Falun are erecting a Helfenstein furnace for iron ore reduction. A Helfenstein furnace with six shafts has a capacity of 24,000 hp and can produce 250 tons of pig iron per day.

Electrical Precipitation of Suspended Particles.

One of the interesting papers presented before the section on electrochemistry was read by **Mr. Linn Bradley,** engineer of the Research Corporation, who gave a brief review of the processes of condensing dust and fume by means of high-tension electric currents. These processes are the invention of Dr. F. G. Cottrell, who has turned over his patent rights to the Research Corporation to be administered in the interests of the Smithsonian Institution. The Cottrell processes have been described in this journal, March, 1912, page 172, and May, 1912, page 313. Mr. Bradley's paper dealt only with the various applications of the process, one of which, in the cement industry, was described in our special Congress issue, page 611.

The problem of removing suspended particles from gases becomes a serious one when the volume of gases is very large, or the temperatures high, or the gases contain materials which exert a corrosive action upon flues, chambers and apparatus.

The variety of problems and conditions is very large, but they may be divided into three general classes, as follows:

1. Where the composition of the gases is such that they

are the cause of complaint on the ground of nuisance and damage to property.

2. Where the values contained in the gases render their recovery quite necessary on the ground of economy, but where the gases do not constitute a nuisance.

3. Where the gases may be considered sufficient cause for complaint, and yet contain values which should be recovered.

PRECIPITATION OF SMOKE AND CINDERS.

In the first class, and as a further illustration of the growing range of application of these processes, may be mentioned the black smoke, ashes and cinders coming from the stacks of some of the large power plants where the boilers are being forced. In some of these plants the operating departments have endeavored in various ways to purify their gases, so as to avoid any complaints. In a small plant, with proper arrangement of boilers, arches, settings and flues, the stack gases seldom give rise to complaint. Undoubtedly the proper method of attacking this problem is to endeavor to obtain proper combustion. However, there are a great many instances where complete combustion is not desired, and in such cases these processes may prove to be of value in overcoming the nuisance due to black smoke or other suspended particles.

Among its other activities the Research Corporation is at present engaged in demonstrating the removal of such suspended particles by the electrical precipitation processes at one of the largest power plants in the City of New York. In this plant there are nearly one hundred boilers, connected to four smokestacks approximately 300 ft. high by 22 ft. in diameter. The average velocity of the gases through these stacks is estimated at 38 linear feet per second. During the evening, when the peak-load is on the station, it is necessary to force the boilers, so that they are delivering perhaps 100 to 150 per cent in excess of their rated capacity. Automatic stokers are used, but even with the greatest care considerable ashes and cinders are caught up by the draft and carried toward the stacks. The cross flues have been enlarged, in order to reduce the velocity of the gases; but, although some of the heavier particles are deposited in these chambers, the exit gases carry a fairly large amount of cinders and ashes.

Experiments upon this particular material indicate that the greater portion of the large particles can be caught if the velocity in the flue is reduced sufficiently and ample time is allowed for the particles to settle out of the draft by gravity; but flues large enough to produce this result would be prohibitive. Some particles are so light that a slight movement of the gas easily keeps them in suspension. If we endeavor to increase their weight by water sprays or similar means, we interfere with the draft, and may not find our efforts successful, because some particles resist moisture and are extremely difficult to remove by this method. If we are able to construct an apparatus which will not be prohibitive in size, and does not seriously interfere with the draft, whereby we can overcome the buoyancy of the particle in the gas and remove it from the influence of the draft, we will succeed in cleaning the gas of the suspended particles.

The actual net amount of energy necessary to withdraw these particles from the moving gas is very small indeed. The problem is to design and construct such an apparatus, then to place the electrical charge upon the particle and introduce sufficient energy to compel the particle to migrate to some point away from the rapidly moving gas. This may be accomplished by the use of high-tension direct current, either intermittent or continuous. Discharge electrodes and collecting electrodes are placed in a chamber and the gas containing the particles passed between them. The gas is ionized and some of the ions collect upon the particles, thereby charging them electrically. Inasmuch as the charged particles are within a strong electric field of constant polarity, they are driven to the collecting electrodes.

The construction of these electrodes may be varied considerably, to suit the special conditions of each particular problem. In most instances the collecting electrodes are grounded, and to meet different conditions various forms of electrodes are used. The voltages used range from 15,000 up to as high as 50,000 or more, depending upon the size of installation and conditions of the gases to be treated. It will be realized that it is desirable to operate on gases at a high velocity and during a short period of time, in order that the installation may be made as small as possible. Results at present in hand indicate the feasibility of removing the cinders, ashes and black smoke from these power house gases at a reasonably high velocity and without any serious interference with the draft.

Much more might be said about this problem, but it may be more to the point, in this general survey, to mention some phases of the work bearing upon other industries.

APPLICATION IN SMELTING ELECTROLYTIC SLIME.

Shortly after the organization of the Research Corporation, last February, investigations were undertaken on the gases arising from the smelting of the slimes obtained in electrolytic copper refining. The copper, as received at the electrolytic refinery, generally contains some gold, silver and other precious metals, as well as arsenic, selenium, tellurium and many other elements in small quantities. During the purification of the copper by electrolysis most of these impurities settle out from the electrolyte. This sediment, called electrolytic slime, is then treated in another department, primarily for the recovery of the gold and silver.

It is common practice in some refineries to dry this slime and then give it a furnace treatment. The volatile elements are driven off in the gaseous state and subsequently condensed in the flues. During the furnace treatment considerable silver is vaporized, some of which condenses quite near the furnace exit. At this point in the flue dust may be found, assaying very high in silver. Considerable gold may also be found at this point, probably having been carried over mechanically. Further along the flue arsenic, antimony, selenium and other elements, or their compounds, condense, thus forming a thick chemical fume.

It is interesting to note the tendencies of the various compounds to deposit. For instance, some of the silver compounds or metallic particles are very light and finely divided, and it is extremely difficult to effect a satisfactory recovery by settling chambers, centrifugal dust apparatus or by water sprays and scrubbers. Gold has been found a long distance from the furnaces, but as a rule it is the first to deposit. The influence of several volatile metals upon the volatility of other metals, especially at various temperatures, has not been studied very extensively, but most authorities state that the loss of silver is increased in the presence of lead or zinc. Probably arsenic or antimony also exert an influence upon silver. It would be interesting and of considerable value to be able to consult curves of vapor tensions of these various elements under the varying conditions of these refinery gases.

Samples of this kind of dust from various refineries show it to have a range of at least from 100 to 2,000 ounces of silver per ton. The gold content is not apt to vary so greatly. Fume has been collected after passing through water scrubbers (so arranged that the gas was saturated and carried water particles in suspension) and found still to contain large amounts of precious metals. The character of the fume will depend, of course, upon the composition of the copper received at the refinery, as well as upon the operations of the refining furnaces. It is almost impossible

to filter this gas, because of its corrosive action upon the filtering medium and its tendency to clog the same. Results obtained upon these gases lead to the conclusion that practically all of the values can be readily recovered by the electrical precipitation processes, thereby insuring a greater net profit in these departments.

The problem of recovering the values from the dust collected is receiving considerable attention. If all the collected dust is fed back into the furnace the highly volatile elements will again be driven off, subsequently condensed, and, in turn, collected. This will permit the impurities to concentrate and increase to such a point as to be objectionable. Experiments are now being conducted with a view to facilitating the recovery of the precious metals in a pure state from this precipitated fume. The proposed method may open up an opportunity for improvements in the present practice of treating these electrolytic slimes.

CONDENSATION OF ACID MIST.

The electrical precipitation processes have been in steady use for the collection of sulphuric acid mist from parting kettles, in one silver refinery for a number of years, and have given excellent results. Another application recently developed is the equipment of acid concentration stills, so as to recover the small amount of sulphuric acid being carried away by the gas. The installation is very compact, the power consumption small, and very little attention is required. For installations of this character special devices for overcoming insulation difficulties and promoting the ionization by the discharge have had to be worked out. The pubescent electrodes are one of the most important features in installations of this type.

PRECIPITATION OF SMELTER FUMES.

One of the large lead smelters in the western portion of the United States is installing the processes on its roaster stack. Godfrey machines and Dwight-Lloyd sintering machines are employed, delivering their gases to a common flue. This flue has a cross-section of such size that the average velocity of the gases is about 500 lin. ft. per minute. A large amount of the very heavy particles is deposited in the first 200 ft. of flue, but the fume is being discharged from the stack after having traveled a much greater distance.

Preliminary tests were made on these gases at a point about 150 ft. from the stack base. The temperatures varied from 100 deg. C. to 13? deg. C. During these preliminary tests the ore charges upon the sintering machines were varied and checked against the assays of the dust recovered electrically, during the day, in order to ascertain how the different char__ influenced the loss of lead. Assays of recovered dust var__ from 20 per cent to 45 per __ lead during this period. Even at this distance from the roasters considerable silver and an ap__ ount of gold was found in each sample. dust recovered was dry and could be handled easily w__ was so dense and compact that a little draft did not blow it away.

The smelter is now proceeding with an installation on a larger scale and it is expected that considerable values will be recovered. Furthermore, this installation may make it possible to push the roaster department to a greater tonnage, due to the ability to collect any materials carried in suspension by the gases. In some roasters, a fairly large percentage of sulphur trioxide is produced. Apparatus can easily be designed to handle this character of gas successfuly, without undue deterioration.

PURIFICATION OF ILLUMINATING GAS.

Another recent application of these processes is in the removal of tar and suspended carbon from illuminating gas. Mr. Walter A. Schmidt has directed the work upon the gas

manufactured from crude petroleum at one of the gas plants near San Francisco. A complete clearance of all visible particles was obtained. In the eastern part of the United States the work of Prof. A. H. White, of the University of Michigan, should be mentioned. Under the auspices of the Michigan Gas Association, Prof. White has conducted extensive investigations, primarily upon the removal of tar from gas manufactured from soft coal. The results were so gratifying that negotiations have been instituted for a larger installation for this purpose. The tar is collected upon the collecting electrodes and runs off into sealed chambers. This field promises to be a very wide one.

Several months ago, prior to the organization of the Research Corporation, Dr. F. G. Cottrell made arrangements with Prof. R. K. Duncan, of the University of Pittsburgh, where the latter arranged to use the Cottrell processes for conducting investigations upon the smoke problem of the city of Pittsburgh. Dr. Strong, Dr. Benner and others have been of assistance in this important problem.

Many other problems of a similar nature to some of those mentioned above have been brought before us and an effort will be made to adapt the processes to them at the earliest possible date. There are many places adjacent to New York City wherein considerable values may be recovered, and at the same time improve the atmosphere, at least in so far as suspended particles and obnoxious fumes are concerned.

Subsequent to the reading of the paper, Mr. Bradley explained in detail the construction of the small apparatus prepared for demonstrating the Cottrell processes. The first test was upon the precipitation of cement dust, which was blown through a pipe 6 inches in diameter. The effect of the high tension current was to collect all of the dust within the pipe. The cement dust was shipped from the plant of the Riverside Portland Cement Co., at Riverside, Cal., having been collected by the Cottrell processes at that plant. The dust was placed in a bottle and compressed air introduced in such a manner as to pick up and distribute the dust within it. A test was next made upon some chemical fumes. Compressed air was blown through bottles containing concentrated ammonia and concentrated hydrochloric acid, separately, and the chemically laden gases mixed in a chamber just prior to being introduced into the electric treater. Dense fumes of ammonium chloride were thus obtained and the effect of turning the current on and off was demonstrated. A further and last test was upon tobacco smoke.

Section on Metallurgy and Mining.

Sampling and Assaying of Silver Ores Containing Cobalt, Nickel and Arsenic.

The ores found in the Cobalt, Ontario (Canada), mining district are arsenides of cobalt, nickel and silver. They contain also native silver, and on account of their complex nature are difficult to sample and assay with accuracy. Consequently, various local methods have been adopted. This paper by **James Otis Handy**, of Pittsburgh, Pa., describes the practice of hand sampling at the Pennsylvania Smelting Company, Carnegie, Pa., and the methods of control assay used in the Pittsburgh Testing Laboratory.

Sampling.—Low grade ore is shipped in bulk, and high grade ore and concentrate in sacks and barrels. The moisture sample is taken when the cars are unloaded or when the sacks and barrels are opened. For low-grade ore the sample consists of four separate portions of 6 lb. each, taken at intervals during unloading. With the high grade the sample is selected from different parts of the pile after it is thoroughly shoveled over.

The principle of the methods of sampling adopted is that of

successive selections of a portion of the ore, with intermediate grinding of the selected portion. Low-grade ore is first crushed to 1½ in. and divided by shoveling two parts to the storage bin and one part to the sampling floor. The floor pile is then divided by shoveling one to bin and one to sample. The latter is now one-sixth of the whole lot and should not weigh less than 6000 lb. This is crushed to ½ in. and quartered to 1700 lb.

The ore is crushed again till all passes the ¼-in. screen, then turned and shoveled over at least twice. It is then quartered down to not less than 300 lb. which is crushed by rolls to pass a 10-mesh screen, then turned and mixed thoroughly and quartered to not less than 60 lb. This sample is ground in a mill to pass 20 mesh and is cut down by a Jones sampler to about 4000 grams, which is ground in a pebble mill till the ore (except metallics) all passes a 120-mesh sieve. Each part of the final sample consists of "pulp" and "metallics" weighing together about 1000 grams per sample. Four sets are made. If the total amount of "metallics" from the 4000 grams is not over 3 grams, it is jointly assayed by the assayers representing buyer and seller. The "metallics" are scorified with test lead and borax glass and the button is cupelled. The weight of the silver bead is then recorded on each sample envelope with the weights of moisture, pulp and "metallics." If the "metallics" exceed 3 grams they are divided into two or more parts, to accompany the pulp samples.

High Grade Ore.—Under this head is included all ore containing over 500 oz. of silver per ton or ore 300-500 oz. containing its metallics in coarse form and unevenly distributed. The method of sampling is subject to some variation depending on the character of the ore, but the usual method follows:

The ore is all crushed to 1-in. size and divided into halves, alternate shovelfuls being taken to the storage bins and to the rolls, which are set to crush to ½ in.

The metallics on the screen are collected, weighed, melted and cast into bars. The bars are weighed, sampled and assayed. The ore passing ½-in. screen is made into two piles of equal weights by alternate shoveling, one man doing the work. Each pile constitutes a sample and is treated separately as follows:

The ore is coned three times in order to mix it thoroughly, then coned and quartered until the weight is about 2400 lb. This amount is crushed by rolls to pass a ¼-in. screen.

This product is coned three times to mix thoroughly, then quartered and coned until reduced in weight to from 400 to 600 lb. The ore is then crushed in rolls to pass 1/10-in. screen. Any metallics are ground in a drug mill to pass the 1/10-in. screen. The sample is then cut down by the use of a Jones sampler to about 75 lb. This is then ground to pass a 1/20-in. screen and reduced by the Jones sampler to about 5 lb. The sample is ground fine in a pebble mill, screened on a 120-mesh screen, giving 120-mesh "metallics" and "pulp."

The other half of the ore (through ½-in.) is sampled in the same manner as described above. Both sets of samples are assayed and results averaged. If the ore is very rich (over 1000 oz. Ag per ton) it is divided into three parts after passing through the ½-in. screen. Each part is then run down separately. Small amounts of metallics on ¼-in. screen are ground up in a drug mill and returned to the sample. Larger amounts are melted into a bar and the bar assayed.

Method of Assaying.—There have been at least three distinctly different methods in use for assaying these arsenical cobalt-silver ores.

(a) The combination of wet and fire assays.
(b) The scorification fire assay.
(c) The crucible fire assay.

The writer has had best success with method "(c)." The manipulations are fewer and simpler than in "(a)" and the loss of silver by volatilization has been found to be more perfectly controlled than in either of the other methods. We know, however, that other assayers use methods "(a)" and

"(b)" with excellent results. We shall describe only our own practice.

The Crucible Fire Assay.

Fluxes (a) silver ore flux:
    Soda ash (Na₂CO₃)...................4 parts
    Pearl ash (K₂CO₃)....................4 parts
    Borax glass (Na₂B₄O₇)...............2 parts.
    (b) Litharge.
    (c) Borax glass for cover.

Assay of Low-Grade Ores.

Fusions are made in a gas-fired muffle furnace (Case type with fan blower attached). "20-gram," "Denver" or "Battersea" crucibles are used. Six assays are run at once with two empty crucibles in front.

Charge:

Pulp ....0.2 A. T.
Silver flux ....30 grams
Litharge ....60 grams
Flour .... 1 gram

Borax glass cover.. 8 grams.

Mix the charge by rolling on a rubber cloth and transfer to crucible. Cover with the borax glass. Place crucibles in muffle at medium red heat. Hold for 20 minutes, then close muffle door and raise temperature to bright yellow and hold for 30 minutes. Pour into conical iron molds and when solidified hammer free from slag.

Cupel on single rows across the muffle. The temperature and draft control are the important factors. The temperature is kept as low as possible and conditions must be freely oxidizing without excessive draft.

The temperature must be low enough at the finish so that litharge feathers form on the cupel around the button. Large silver buttons after the "blick" are covered with hot cupels. Small buttons are simply cooled slowly to avoid "sprouting."

"Metallics" are run separately by scorification, using test lead with borax glass and silica flux (equal parts borax glass and sand, both ground fine). Temperature control both in scorification and cupellation are vital.

"Silver bars" from nuggets or coarse "metallics" are sampled by drilling top and bottom. The drillings are assayed for fineness. Three hundred milligrams are wrapped in pure sheet lead and cupelled with a pure silver proof beside it. The weight of silver taken for the proof is 85 to 90 per cent of the weight of the drillings of the sample (250 to 270 m.g.). The percentage loss of silver suffered by the cupelled proof is added to the silver recovered by cupellation of the drillings.

Assay of High-Grade Ores.

Charge:

Pulp .... 0.2 A. T.
Silver flux ....30 grams
Litharge .... 90 grams
Flour .... none. (The ore is a reducer.)

Borax glass cover.. 8 grams.

Fusion is made as in the case of low-grade ores. The larger quantities of "metallics" from high-grade ores are best assayed by the crucible process rather than by scorification.

Charge:

Metallics .... 0.5 gram
High-grade ore flux.................50 grams
Flour .... 1 gram

Calculation of Results.

Let x = total weight of sample (a) pulp; (b) metallics
" y = assay of pulp (ounces Ag per ton of pulp)
" z = milligrams of Ag in total weight of metallics.

Then $\dfrac{(a)}{x}$ = per cent pulp, and $\dfrac{(b)}{x}$ = per cent metallics.

Then $\dfrac{x}{29.166}$ = A.T. (assay tons) of sample.

Then (1) $y \times$ per cent pulp $=$ ounces Ag per ton of ore (from pulp sample).

Then (2) $\dfrac{z}{\text{A.T. of sample}}$ $=$ ounces Ag per ton of ore (from metallics.)

(1) plus (2) $=$ ounces Ag per ton of ore (total).

If there is a bar of silver made from coarse metallics or nuggets, it is usually paid for on the basis of its weight and assay value. If the total original assay value of the ore is to be calculated, it must include the Ag in coarse metallics.

This involves the factors:

Per cent of coarse metallics or nuggets.
Weight of bar from nuggets.
Per cent of screenings from nuggets.

The calculation is easily made.

Agreement Between Assayers.

The following figures illustrate how well independent assayers agree as to the value of samples of these complex ores prepared by the methods described.

	Mark	P.T.L.	P. S. Co.
37396	Seconds		
	Pulp	74.25 ounces	73.53 oz. Ag per ton
	Metallics	61.96 "	62.79 "
	Total	136.21 "	136.32 "
	Fines		
	Pulp	114.23 ounces	114.42 ounces
	Metallics	28.23 "	28.22 "
	Total	142.46 "	142.64 "
37691	Total	54.07 "	53.79 "
37252	L.G.X.		
	Pulp	1225.85	1225.00
	Metallics	1295.53	1293.95
	Total	2521.38	2518.95
37252	H.G.		
	Pulp	503.17	502.14
	Metallics	558.34	559.24
	Total	1061.51	1061.38

These agreements are exceptionally close and represent a high degree of skill in manipulation acquired by the operators, especially in the close control of cupellation losses.

The methods of sampling and assaying are almost entirely those worked out by the Pennsylvania Smelting Company. Their accuracy has been confirmed by repeated re-sampling and re-assaying.

Copper-Zinc Alloys.—A paper by **Wilder D. Bancroft** and **J. M. Lohr** of Cornell University, gives the results of an investigation made to establish the relationship between the constitution of the copper-zinc alloys, and the tensile strength of the cast metal, unworked and without heat-treatment.

Brasses containing 100 to 47.5 per cent copper were tested. The microscopic examination showed that the alpha-phase exists as far as 66 per cent copper. The alpha + beta phase extends from this point to about 57 per cent copper, while the pure beta-phase occurs between 57 and 50 per cent copper.

The changes in the tensile strengths of the alloys agree very well with the changes in the internal structure as brought out microscopically. The values remain pretty uniform throughout the alpha-field and increase very rapidly when the beta-phase begins to appear at about 66 per cent copper. As the proportion of pure beta metal increases the strength continues to increase until a maximum is reached in the field for pure beta. When the brittle gamma metal makes its appearance the strengths decrease rapidly.

The maximum tensile strength occurs in the neighborhood of a 55 per cent copper alloy, and its value is about 71,000 pounds per square inch.

"It is possible to obtain a cast brass having an ultimate tensile strength of 71,000 pounds per square inch, and an ultimate elongation of 14.8 per cent; or a brass having an ultimate tensile strength of over 36,000 pounds per square inch and an ultimate elongation of 35.6 per cent."

Adsorption of Coloring Matter.—A paper in French by **A. Bouchonnet,** of Paris, France, deals with the adsorption of coloring matter of ochres. The author investigates the factors which can accelerate or retard the adsorption, namely, temperature, concentration, acidity, etc.

Errors in Gas Analysis Resulting from the Assumption that the Molecular Volumes of All Gases Are Alike.—A paper by **George A. Burrell** and **Frank M. Seibert,** of the Bureau of Mines, Washington, D. C.

The authors submit actual calculations from gas analyses they have performed in which is shown the need for correcting well-known equations so that the errors which result from the assumption that the molecular volumes of all gases are alike will not occur.

The best density determinations made of the common gases were chosen and the reactions for complete combustion with oxygen of carbon monoxide, methane, ethane, propane, etc., were changed accordingly.

The combustible constituents in natural gases are changed in some cases as much as 2 per cent when the correct equations are used.

The application of the study consists in the use of the corrected equations when analyses are made of gas mixtures containing high percentages of combustible gases.

Section on Fuels and Asphalts.

Neutral Coke.—A paper by **J. R. Campbell** of Scottdale, Pa., gives details of a neutral coke test made at the Gary coke plant in order to investigate the claims of the Diehl-Faber process for making neutral coke. In this process, by the introduction of limestone into the charge in certain proportions, it is claimed that the sulphur in the raw coal is transformed into calcium sulphide during the coking process, and that the physical quality of the resultant coke is improved. The effect the limestone will have upon certain by-products, i. e., the gases and the ammonia yield, was also investigated; also the temperature conditions that must prevail to secure the desired results. The results of these tests are given in tables or diagrams and the final conclusions are as follows:

The temperature of the coking mass is insufficient for the complete transformation of S in the coal to CaS in the coke during the coking process, and such temperature must be over 2100 deg. Fahr., perhaps nearer 2500 deg. Fahr.

The temperature is insufficient for the complete formation of lime silicates.

It is not practicable to use more limestone than will combine with the ash of the coal to form a mono-silicate.

The physical quality of the coke is improved by the limestone additions if too much free CaO does not remain in the coke and if such additions are for the formation of a mono-silicate, as evidenced by the physical tests of the coke and its physical appearance, provided always that high temperatures prevail at or near the close of the coking process.

The decomposition of the limestone and the action of the resultant gas on the incandescent coke, forms high percentages of CO gas, which lowers the B.t.u. value of the whole gas about 10 per cent.

The coking time will be extended by the introduction of limestone into the charge, if any beneficial results are to be expected.

The total NH₃ yield will be increased quite materially as evidenced by bomb tests.

To attain sufficiently high temperature in the coking mass for the desired chemical reaction and permanent physical qualities in the coke, will necessitate maintaining the flues at critical temperatres say: 2700 deg. F. to 3000 deg. F.

The Diehl-Faber process has but little practical value from a blast-furnace standpoint, unless all the sulphur is converted to CaS during the coking process, or unless a silicate is formed, for the reasons that S other than CaS will enter the iron readily and that the free CaO will cause the coke to crumble upon exposure to the atmosphere.

The results of bomb tests are given in a separate report. The conclusions are:

If "neutral coke" were made from washed Rend coal the gas and sulphate yields would be increased, per ton of actual coal charged, due to well understood chemical reactions.

The percentage of combined NH_3 is radically decreased by the limestone additions.

While the volume of gas is increased by the limestone additions, the B.t.u. value is decreased; consequently, there will be a smaller percentage of available gas.

Purifying Coal Gas with Recovery of By-products.

—A paper by **Karl Burkheiser** of Hamburg, Germany, describes the Burkheiser wet desulphuretted hydrogen, cyanogen, and ammonia purifying and recovery process.

The object of the Burkheiser process is to absolutely purify coal gas and to combine the sulphur contained therein with the ammonia so as to form ammonia salt. In the development of his process the author started from the principles used up to now for the sulphuretted hydrogen purifying by the use of bog ore. But he submits the bog ore before its use to a preparation by roasting, and thereby removing all the organic parts, and also by withdrawing from it part of the chemically combined waters. From the original bog ore $(Fe_2(OH)_6)$ is formed a red modification, poorer in water (probably from the combination $Fe_2O_3H_2O$), which has a great absorbing capacity for the sulphuretted hydrogen. By so doing it is not only possible to confine the gas purifying itself to the smallest imaginable space, but also to give the purifying mass a form which gives, in the oxidation of the iron sulphide combinations to SO_2 and SO_3, an absolutely pure SO_2 and SO_3.

The fundamental principles of the process consist, in short, of completely removing, by means of the above-mentioned purifying mass, the sulphuretted hydrogen from the gases, and in the regeneration of this purifying mass to drive the oxidation up to sulphur dioxide and sulphur trioxide, and then to wash out by one and the same wash fluid the so-formed sulphurous acids as well as the ammonia of the gases without the carriers of these two products coming in contact.

The paper describes the different steps of the process in some detail.

Refining Light Oil.

—A paper on "coal tar light oil in the United States: the manufacture, nature and uses of products derived therefrom," by **John Morris Weiss**, of New York, N. Y., gives a summary of the present status of refining light oil in the United States.

Up to the present the manufacture of products in this field has been mainly confined to materials already existent in the crude oil, and capable of being isolated by physical and simple chemical means. The further, more intricately chemical manufacture of the various derivatives of benzene, toluene, naphthalene, etc., which serve as raw materials for dye manufacture, has as yet been only carried on in a comparatively small way in the United States.

The chemical possibilities along these lines are yet to be developed in this country, and here, indeed, is a wide field for technical endeavor which will place the coal tar industry of the United States in a stage of development equal to that already attained abroad.

Coal Preparation.

—A paper by **W. S. Ayres,** of Hazleton, Pa., discusses "the technical problems of coal preparation." The preparation of coal, whether anthracite, bituminous or other lignite, or any of the grades between or beyond them, must make use of the same laws, namely, those governing the difference in specific gravity and those governing the difference in the angle of repose. One of these groups may be perfectly efficient in preparing one coal having certain physical characteristics, while for another coal the other group would do perfect work, and for almost infinite mergings of the two typical classes of coal a series of combinations and modifications of the two principal methods would have to be employed.

In the specific gravity process of preparation, typified in the jig, many perplexing problems have arisen and many annoying results are always present, depending for their degree of seriousness upon the physical nature of the material being treated.

Of the other mechanical devices for removing impurities from coal there are three that are extensively used, each employing the difference in the angle of repose between slate and coal as the basic principle of its operation. These three systems are described.

The most important of all features in the entire process of anthracite preparation is to so handle the coal in transportation, screening, separation, conveying and loading as to retain the pieces of coal whether coming directly from the mine or from the crushing-rolls in their initial form without any loss in weight from abrasion or chipping, if at all possible to do so. This must be taken into consideration in the transportation and loading of the coal, etc.

Low-Grade Lignite Coals.

—A paper by **E. J. Babcock** of the University of North Dakota, gives an account of extended "investigations relative to the better utilization of non-coking low-grade lignitic coals."

There are enormous Western lignite deposits. North Dakota alone has a coal area of over 32,000 sq. miles, capable of producing 400 or 500 billion tons.

The author's extended researches indicate that the concentration of the raw lignite into a fuel of relatively high heating values depends upon, first, the removal of the moisture; second, the expulsion and saving of the volatile gas, and, third, the binding together of the concentrated residue into a strong, durable, and satisfactory briquetted product. The best utilization of these lignites is therefore a combination of briquetting with gas production, while in addition there is a considerable amount of ammonium sulphate formed (nearly 15 lb. per ton of dry lignite) and tar, both of which have a ready market.

One and one-half tons of lignite will produce about a ton of these briquets in addition to 12,000 or more cubic feet of gas. The briquets have above twelve-thirteenths the actual heating value of average anthracite and can be made at a sufficiently low cost to put the industry on a commercially satisfactory basis in at least some of our Western states.

Considering the ease with which the gas is produced, the low price of the original lignite, and the value of the residue, this gas should have a large commercial value for heating, lighting, and power purposes because of the low price for which it could be sold if manufactured in a plant used to produce briquets from the residue.

The Constitution and Genesis of Certain Lignites and Sub-Bituminous Coals.

—A paper by **Reinhardt Thiessen,** of the Bureau of Mines, Washington, D. C. Coals are composed chiefly of a residue of the most resistant components of plants, of which resins, resin-waxes, waxes and higher fats and alcohols are the most important.

Plant substances differ very markedly in their resistance to various agencies. These substances involved in the life and the support of the plant are relatively labile, whereas those involved in some protection function, or are to be looked upon as waste products, are relatively stable. Of these substances, the resins, waxes, resin-waxes, higher fats and alcohols are the most stable.

After the death of the plant there begins, governed by various conditions imposed upon the deposit, a partial decomposition, maceration, elimination and chemical reduction, brought about chiefly by the agencies of organisms, fungi

first, followed closely by bacteria. The least resistant components are removed first, leaving the more and the most resistant behind in a residue called peat.

The process of elimination and chemical reduction begun in the peats, chiefly by biochemical means, is taken up and continued by dynamochemical means into and through the various grades of coals.

Lignites and sub-bituminous coals are therefore composed of a residue or debris consisting of the more and the most resistant plant substances in a macerated and changed condition, in which are usually embedded logs, stems, branches, or fragments thereof, also in a much changed condition, in varying proportions.

The "wood" has been reduced very much, and since its resin contents have suffered little if any change, it has become very resinous.

The debris is composed (a) of cellulosic substances in a greatly altered condition, in the shape of fragments of tissue, small complex of cells, single cells and fragments of cells; (b) resinous substances; (c) cuticles; (d) spore-exines; (e) pollen-exines, and (f) a variety of a small number of other unimportant bodies. The interstices of these bodies are in turn filled in with a still finer originally macerated matrix consisting essentially of the same substances, either in a very high state of comminution or the residue of them. This component is termed the ground or binding mass.

Highly macerated leaf cuticles are an important component of the ground mass.

The proportion, or distribution, of the various bodies named, varies considerably in different beds and in different layers of the same beds.

The ratio of the cellulosic components to the resinous and cuticular components varies, generally, inversely to the state of maceration and the degree of metaphorism of the coal.

Section on Physical Chemistry

Temperature Scale.—A paper by **George K. Burgess,** of the Bureau of Standards, Washington, D. C., on "the present status of the temperature scale," discusses the thermodynamic scale, the gas scales, secondary methods and scales, and the calibration of the platinum resistance thermometer and the thermoelectric thermometer. The following list gives standard temperatures in the thermodynamic scale. Boiling points are for a pressure of 760 mm. Hg.

STANDARD TEMPERATURES. THERMODYNAMIC SCALE.

Substance	Phenomenon	Temperature C°	Uncertainty C°	Reproducibility C°
Hydrogen	Boiling	−252.7	0.2	0.05
Oxygen	Boiling	−182.9	0.1	0.03
Carbon dioxide	Sublimation in gasolene	−78.34	0.1	0.03
Mercury	Freezing	−37.7	0.1	0.05
Water	Freezing	0	0	0.001
Na$_2$SO$_4$+10H$_2$O	Transformation to anhydrous salt	32.383	0.002	0.001
Water	Boiling	100	0	
Naphthalene	Boiling	271.96	0.02	0.01
Tin	Freezing	231.85	0.1	0.05
Benzophenone	Boiling	305.90	0.05	0.02
Cadmium	Freezing	320.92	0.1	0.03
Lead	Freezing	327.4	0.1	0.05
Zinc	Freezing	491.4	0.1	0.15
Sulphur	Boiling	44.46	0.1	0.03
Antimony	Freezing	630.	0.5	0.3
Ag^2-Cu^3	Eutectic freezing	779	1.0	1.0
NaCl	Freezing	800	2.0	1.0
Silver	Freezing	960.5	1.0	0.5
Gold	Freezing	1063	2.0	1.0
Copper	Freezing	1083	2.0	1.0
Palladium	Freezing	1549	10	3
Platinum	Melting	1755	15	5
Alumina	Melting	2000	30	20
Tungsten	Melting	3000	100	25
Carbon arc	Pos. crater	3600	150	50
Sun	Surface	6000	500	100

For the temperatures dependent on atmospheric pressures we have:

Oxygen B.P. $T=T^{760}+0.013\ (p-760)$
Carbon dioxide S.P. $T=T^{760}+0.017\ (p-760)$
Water B.P. $T=T^{760}+0.037\ (p-760)$
Naphthalene B.P. $T=T^{760}+0.058\ (p-760)$
Benzophenone B.P. $T=T^{760}+0.063\ (p-760)$
Sulphur B.P. $T=T^{760}+0.0912\ (p-760)-0.000042\ (p-760)^2$

As to the chemical purity of the various substances, Mylius has shown that the following metals may easily be had to 0.01 per cent or better: Au, Ag, Pt, Hg, Cu, Sn, Pb, Cd, Zn. Of the boiling substances, benzophenone and oxygen are the only ones to the purity of which special attention need be given.

To show how carefully these standard preparations have been determined, the author gives in detail the results of all measurements since 1900 on some of the most carefully studied of these temperatures, such as the boiling points of oxygen and sulphur and the melting points of cadmium, gold, platinum, and tungsten.

Of course, an international agreement on the temperature scale is of fundamental importance. In pursuance of this idea, the several national laboratories and the International Bureau are at present interchanging communications relative to the establishment by agreement of some such single temperature scale, and it is to be hoped that the outcome of these interchanges will be of great practical benefit and convenience for the certification and use of temperature-measuring instruments, and that thenceforth 500 deg. C. or 1000 deg. C., for example, will each convey but one idea of temperature as do 0 deg. or 100 deg. C., at the present time.

Melting Points of Fire Bricks.—In a paper by **C. W. Kanolt,** of the Bureau of Standards, Washington, D. C., an account is given of the determination of the melting points of 54 samples of fire brick, including fire clay, bauxite, silica, magnesia, and chromite brick. The determinations were made in an electric vacuum furnace, the temperature being measured with an optical pyrometer.

The following melting points of materials important in the manufacture of fire brick were determined:

Kaolin ....1740° C.
Pure alumina ....2010° C.
Pure silica ....1750° C.
Bauxite ....1820° C.
Bauxite clay ....1795° C.
Chromite ....2180° C.

An improved method of calibrating the Holborn-Kurlbaum optical pyrometer is described.

Allotropy of Iron and Silver Iodide.—A paper by **Carl Benedicks,** of the University of Stockholm, Sweden, on the allotropy of iron and silver iodide points out that for the metallography of the iron, its allotropy is still the most vital question. The results of the author's investigation are as follows:

The very common opinion that allotropy means a quite sudden transformation from the one modification to the other cannot, a priori, be maintained except as a special case (Type I). There is always the possibility that the one modification (Type II, a and b) or both (Type III) have a certain solubility for the other—this is the simplest expression for the fact that the transformation partly takes place continuously. If the solubility were unlimited one could not speak of allotropic transformation point, but allotropic modifications can nevertheless be said to occur. (Type IV.) Which of the types of allotropy (theoretically) possible, actually occur in each case, must be decided by experimental study of one, preferably more, of the properties in the proximity of and at a sufficient distance from the transition point in question.

An experimental study of the dilatation of silver iodide, extended to − 180 deg., has shown that the transformation point of this body at about + 147 deg. is an illustration of Type II, a. At low temperatures the silver iodide has a positive dilatation; only at 0 deg. or room-temperature the well-known negative

dilatation occurs, that is peculiar to silver iodide. The dilatation must be ascribed to a solubility increasing with the temperature in the hexagonal modification of the regular modification stable above the transition point, a solubility which, according to the curve of dilatation, is in all probability greatly limited.

The supposition of a certain solubility has earlier been made by Tammann, but for want of clear proofs rejected by Roozeboom; determinations of dilatation at higher temperatures have earlier been made by Rodwell, and though only used for the position of the transition-temperature, they coincide very well with those of the author.

The simple dilatation apparatus used by the author and with which temperatures between — 180 deg and o deg. could be very conveniently obtained, will probably be of some use on other occasions.

A detailed determination of our physical knowledge of the critical transformation point A, of the iron at about 890 deg. and the lower, more undecided transformation range A$_2$, shows that likewise the transformation of the iron at 890 deg. belongs to Type II, a, of which we thus know two examples. According to this the nature of the so-called beta-iron is to be alpha-iron (stable at low temperatures) containing in solution a limited amount of gamma-iron, increasing with the temperature (stable at temperatures above 890 deg.).

"The opinion is greatly a returning to the original views of Osmond.

"The standpoint now attained simplifies in a high degree our opinion as to the metallography of iron; this is evident from, amongst other things, that the three theories, now acknowledged as the most probable ones, for the constitution of martensite, when seen from this point of view, become identical."

Magnetic Theory.—A theoretical paper by **Jacob Kunz,** of the University of Illinois, deals with the present theory of magnetism and the periodic system of chemical elements. After an introduction on the fundamental assumptions of the electromic theory in its relation to magnetism, the author sums up the experimental facts with special reference to diamagnetism and additive and non-additive properties. He then discusses the periodic system of the elements and their magnetic properties, and further deals with ferro-magnetism and the magnetom by Weiss and the degrees of freedom.

Synopsis of Metallurgical and Chemical Literature

Gold and Silver.

Role of Alumina in Copper Slag Formation.—The question of whether alumina acts as an acid or base in slag formation has never been satisfactorily settled in the minds of metallurgists. In his presidential address at the last (June, 1912) meeting of the Australasian Institute of Mining Engineers, MR. H. C. BELLINGER gave a number of examples of the varied manner in which alumina acts under different conditions. At Rossland, B. C., where the gangue contained 14 to 18 per cent alumina, 10 to 12 per cent lime, 2 to 3 per cent potash and soda, and 40 to 46 per cent silica, one metallurgist regarded the alumina as an acid, and fluxed with limestone. His furnace promptly froze; whereupon, assuming that he had not added enough base, he put more lime on the charge, with worse results. Another metallurgist fluxed his charge on the assumption that the alumina in combination with silica would act as a base, and the balance as an acid. This was found to give satisfactory results, and has since been the basis of operations.

In another case at Butte, where it was required to form a slag containing 45 to 50 per cent silica, all of the alumina (8 per cent) acted as a base. Similar experience is cited at the Northport and Britannia smelters. Another instance is cited in which it was difficult to decide which part alumina played.

A copper ore contained 40 to 45 per cent barite and 8 per cent zinc. It was necessary to add both silica and alumina, and the resulting slag carried 5 to 12 per cent alumina. All slags resulting from various mixtures were satisfactory, and in some cases the alumina undoubtedly acted as an acid.

Mr. Bellinger concludes, as a result of his wide experience, that mere chemical analysis will not serve to reveal the rôle played by alumina. He cites the diversity in composition of the felspars and mentions their different behavior under similar treatment. Further variations are cited among the meta-, ortho- and hydro-silicates, with the range of oxygen ratios from less than 1:1 in the sub-silicates to 4:1 in the bisilicates. "It is when one considers the many conditions of combinations in aluminous compounds, and has noted the definite changes in the action of the alumina in sympathy with variations in such combinations, that it becomes quite feasible
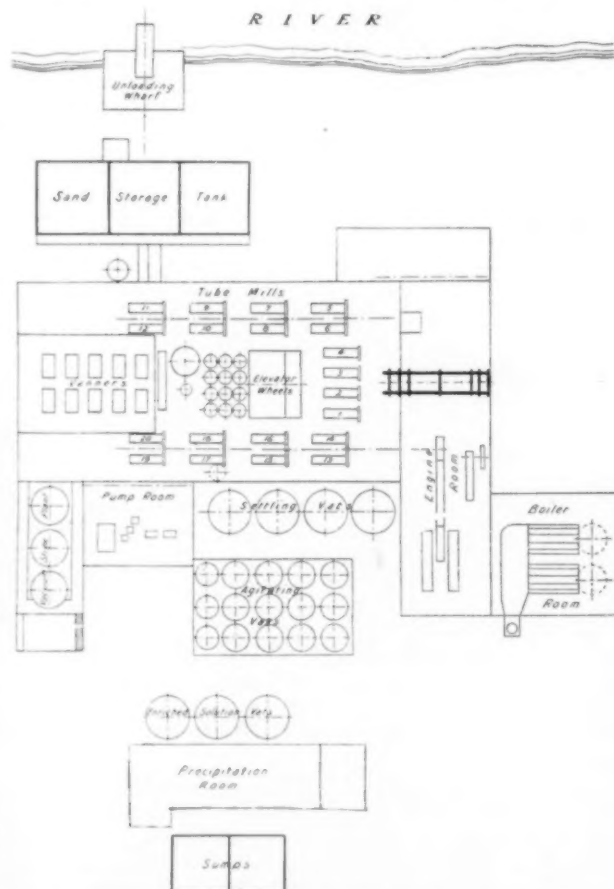

FIG. 1.—GENERAL PLAN OF THE WAIHI-PAEROA GOLD EXTRACTION COMPANY'S TAILING PLANT.

that, under similar furnace conditions, we could reasonably expect the compounds to adopt rôles suited to their peculiar forms."

The author cites the fact that under some conditions alumina has been satisfactorily handled by calculating as acid that portion which is soluble in acids, and the insoluble portion as base or as an independent elementary magma.

Cyaniding Tailing.—With improvement in metallurgical methods, instances of re-treating old tailings are multiplying. In the August, 1912, issue of the *Mining Magazine*, Mr. CHARLES A. BANKS describes a plant which he erected for the Waihi-Paeroa Gold Extraction Co., of Auckland, N. Z., to cyanide 500 tons per day of quartz tailing which had accumulated for the past 20 years in the lower reaches of the Ohinemuri River.

In Fig. 1 is given a general plan of the mill. A dredging plant is used to raise the tailing from the river by an air-lift

of 100 tons per hour capacity. The sand thus elevated is transported by barges to the unloading wharf, where it is screened to remove coarse rubbish and then stored in concrete bins of 3000 tons capacity.

The treatment begins by separating the light river sand and charcoal from the valuable quartz. This is accomplished on vanners, the quartz passing over the head of the machine in a layer about ¼ in. deep, where it is washed off by a weak cyanide solution. The quartz is then ground in tube mills, the tube discharge being elevated to a classifier by means of two tailing wheels having a combined lift of 35 ft. The fine overflow from the classifier is all minus 200-mesh, and is run to settling tanks. The coarse underflow is returned to the tube mills.

The slime is settled with lime and then agitated by air in cone-bottom agitators, 50 ft. high and 13 ft. diameter. The period of agitation is 48 hours. After agitation the slime is filtered under a vacuum of 18 in., forming a cake 1½ in. thick weighing 9 tons, in about 15 minutes. The cake is washed for 18 minutes and then discharged by compressed air. One crane with one pulp and two washing vats 20 ft. diameter, and two filter baskets each of 1660 sq. ft. area, can easily treat 500 tons in 24 hours. Zinc shavings are used for precipitation.

It will be noticed in the general plan of the mill that the tubes are placed around the tailing wheels. This convenient arrangement is made possible by a new method of driving sixteen of the tubes, arranged in two sets of two rows each. A main shaft runs between each pair of rows. Four pinions on each shaft are keyed to friction clutch sleeves, and may be engaged or disengaged quickly and satisfactorily. Each pinion engages two tube mill spur wheels. As the tubes are placed on sole plates with adjusting screws, any one of them may be run out of gear when necessary without interfering with the work of the other.

Mr. Banks has found this method of driving superior to the use of tight-and-loose pulleys and cites the following advantages: (1) Smaller building; (2) mills closer to elevator wheels, and therefore less height to elevate the pulp; (3) saving in capital expenditure and operating cost; (4) no belts; (5) main shafts on ground instead of up on trestles; (6) a balance pinion, the pinion practically floating as it works.

The tubes are 16 ft. long and 4 ft. diameter, run at 27 r.p.m., and require 27 hp. when carrying a load of 4 tons of pebbles. Their capacity is 25 tons per tube in 24 hours, grinding to 200-mesh.

The average assay value of the material treated is 11s. 6d. ($2.75), and it is expected that the tailing will be reduced to a value of only 2s. 6d. ($0.60), with an operating cost of not more than 3s. 3d. ($0.80) per ton, which will leave a margin of profit of about $1.35 per ton. The mill cost about $340,000. No statement is made of the amount of river tailing available.

Cyaniding Concentrate.—A series of tests was made to determine the advisability of cyaniding raw concentrate instead of shipping it to a custom smelter. The results are communicated by Robert Linton in the July, 1912 *Journal* of the Chem. Met. & Min. Soc., of S. Africa.

The analysis of the concentrate was as follows:

Insoluble	44.00	per cent
Fe	26.50	"
Cu	1.30	"
Zn	1.20	"
MnO_2	0.41	"
Al_2O_3	1.40	"
CaO	1.75	"
MgO	0.47	"
As	0.03	"
S	18.20	"
Ag	4.29	"
Au	0.06	"

The silver occurred as argentite and the gold was contained in the silver sulphide. The concentrate was ground to 200-mesh and agitated with cyanide solution in a small Pachuca tank. The first test showed a constantly increasing alkalinity in the solutions used, although no lime was added after commencing agitation. An analysis of the solution after agitating five hours showed no alkaline sulphides, although K_2S probably formed and immediately combined with KCN to form KCNS. This would seem to explain the increase in alkalinity observed and to indicate that the increase in alkalinity involved a direct loss of cyanide.

Tests to see whether lead acetate would reduce the cyanide loss showed that with 1 per cent, or over, of acetate, the alkalinity gradually disappeared, but the consumption of cyanide greatly increased.

The results of all the tests seemed to indicate that the most economical treatment was to grind the concentrate in a solution of about 0.2 per cent KCN, then agitate in a 0.6 per cent KCN solution, raising to about 0.8 per cent KCN toward the end of the treatment. A high protective alkalinity seemed to be of no advantage, about 0.1 to 0.15 being sufficient. Lead acetate seemed to be of no value in preventing the destruction of cyanide, although the use of a small amount served to keep the solution clear.

Copper.

Electrolytic Copper Refining.—In the course of a report on the 1912 meeting of the Australasian Institute of Mining Engineers, the *Mining and Engineering Review* (Australia), for July 5, gives the following account of refining operations at the plant of the Electrolytic Smelting & Refining Company, Port Kembla, N. S. W.

The tank house building is constructed entirely of wood above the cellar level. The tanks number 600 and are 10 ft. long, 3 ft. 6 in. wide and 3 ft. 7 in. deep. They are built of Oregon lumber and lined with 6-lb. lead. The arrangements of tanks is in sections of 30 and 60, set 10 wide and 3 deep, the lowest tank in the series being 7 ft. above the cellar floor. Pohle air-lifts and hard-lead centrifugal pumps handle the electrolyte. The heater boxes are equipped with hard-lead pipes carrying exhaust steam. Live steam is used to make up loss by evaporation.

The desired composition of the electrolyte is 14 per cent $CuSO_4$, $5H_2O$ and 12 per cent H_2SO_4, with a sp. gr. of 1.18 and a temperature of 130 deg. F.

The anodes are brought to the tank house on racks, spaced 5-in. centers as in the tanks. By means of a crane and lifting hooks, a full tank load of twenty-one anodes is lifted at one time and placed in the tank. A similar rack is used for lifting the twenty-two cathodes from the tank. The tank house has a capacity of about 2000 tons of copper per month.

About 14 per cent of anode scrap is made. The current density is from 16 to 18 amp per sq. ft. of anode surface. The slime containing gold and silver is sluiced from the bottom of the tanks into boxes, and after being screened through a 30-mesh sieve, is blown through lead pipes to the refinery.

Thirty tanks are in use for making starting sheets. These weigh about 7 lb. each. About 20 per cent of the sheets are sheared into strips 4 in. wide, to form lugs for the remainder.

The refining of the gold-silver slime is done in lead-lined agitators 5 ft. high and 5 ft. diameter, fitted with hard-lead propellers. As received from the tank house, the slime contains from 30 to 40 per cent copper and is agitated with sulphuric acid and leached until it contains less than 1 per cent copper before being discharged. The slime is then washed, dried and melted into doré bars for electrolytic refining.

Ore Roasting.

Decomposition of Metallic Sulphates.—An investigation in this subject has been made by **H. O. Hofman** and **W. Wanjukow** of the Massachusetts Institute of Technology, Boston,

and is published in the September, 1912 *Bulletin* of the American Institute of Mining Engineers.

Most metallic sulphides, when given an oxidizing-roast, pass through the state of sulphate before they are converted into oxides. All metallic sulphates are completely decomposed by heat, with the exception of those of lead and bismuth, but the temperatures at which decompositions take place vary with the metals. Some metallic sulphates are converted directly into oxides, others first form basic salts. Alkali-earth sulphates show a behavior similar to that of metallic sulphates, while alkali sulphates do not part with their sulphur trioxide even at elevated roasting temperatures.

There are four methods of making these determinations. 1. Heating in an evacuated tube; 2, heating in a tube closed at one end, without air current; 3, heating in a crucible with air-current; and 4, heating in an open tube with air-current.

The results obtained in the first three methods are not exactly comparable to the conditions existing in the roasting of the ores. The authors, therefore, have used the fourth method, as it resembles more closely than any other the roasting operation on a large scale. The substance to be tested is placed in a boat inserted into a horizontal tube impervious to gases; the thermo-couple is held near the boat or the substance; the tube is heated gradually, usually by electrical means; a slow current of purified and dried air is blown through and carries away the gaseous products of decomposition; the latter can be tested chemically whenever this seems desirable.

One manner of carrying out this process is to make first a blank test, using the amperage necessary to reach a desired temperature, with the thermo-couple held in the boat, but out of contact with the substance; to record the rise of temperature for given intervals of time, 30 or 60 seconds, and to plot the data which will furnish a smooth curve. The next step is to charge the boat, raise the temperature in the same manner as with the empty boat, record the rise in temperature for the same intervals of time, and to plot the data. Any deviation of the second curve from the first will show that some change has taken place; there will have been a transformation if the weight of the substance has remained constant, a chemical change if the weight has increased or decreased.

The plan of the investigation was to determine the dissociation-temperatures of anhydrous metallic sulphates, the formation of basic sulphates, the characters of the resulting products, and the speeds of reaction at different temperatures, in order to obtain data applicable to roasting operations.

The furnace used for heating was an electric resistance type furnace similar to the well-known Heraeus furnace employed by Hofman and Mostowitsch in their work on calcium sulphate. The current used was 110 volts. It passed through the ammeter and two resistance coils, one for rough and the other for close regulation. With all the resistances turned on the current was 2 amp.; when all turned off the strength was 18 amp. and the latter sufficed to attain a temperature of 1150 deg. C.

The authors give a detailed record of the results obtained in testing a large number of sulphates. The amount of heat necessary to decompose the several anhydrous sulphates vary greatly. Some are dissociated at low temperatures and others at high. The results are assembled in tabl I. in which they are arranged in the order in which they are dissociated with increasing temperatures.

Considering the formation of basic salt from the point of view of the periodic system it appears that in each group the tendency to form basic salts increases with the rise 'of the atomic weight. Independently of the periodic system the following characteristics are noted:

Normal sulphates which do not form basic salts upon dissociation show both SO_2 and SO_3 in the gaseous product; normal sulphates which form basic salts upon dissociation evolve only SO_3 when they pass from the normal to the basic state basic

Desulphatization of Anhydrous Metallic Sulphates.

Metallic Sulphates.	Temperature of Beginning of Decomposition.	Temperature of Energetic Decomposition.	Products of Decomposition.	Remarks.
	Degrees C.	Degrees C.		
FeSO4	167	480	Fe2O3.2SO3	Yellow-brown.
Fe2O3.2SO3	492	560	Fe2O3	Red.
Bi2(SO4)3	570	639	5Bi2O3.4(SO3)3	White.
Al2(SO4)3	590	639	Al2O3	White.
PbSO4	637	705	6PbO.5SO3	White.
CuSO4	653	670	2CuO.SO3	Orange-color.
MnSO4	699	790	Mn3O4	Dark red to black.
ZnSO4	702	720	3ZnO.2SO3	White, cold and not.
2CuO.SO3	702	736	CuO	Black.
NiSO4	702	764	NiO	Brownish green.
CoSO4	720	770	CoO	Brown to black.
3ZnO.2SO3	755	767	ZnO	Hot yellow, cold white.
CdSO4	827	846	5CdO.SO3	White.
5Bi2O3.4(SO3)2	870	890	Bi2O3(?)	Yellow.
5CdO.SO3	878	890	CdO	Black.
MgSO4	890	972	MgO	White.
Ag2SO4	917	925	Ag	Silver white.
6PbO.5SO3	952	962	2PbO2.SO3(?)	White to yellow.
CaSO4	1,200	...	CaO	White.
BaSO4	1,510	...	BaO	White.

sulphates upon dissociation into MO set free only SO_3; metallic sulphates with bases M_2O_3 and M_2O do not form basic sulphates upon dissociation; the former set free SO_3, the latter SO_2.

Mill Practice.

Elevator Buckets.—The wear on elevator buckets is a considerable item of expense in the operation of concentrating mills. Mr. H. J. Maguire, of Wallace, Idaho, has studied the life and service of elevator buckets and has endeavored to devise an adjustable lip which will be of material advantage. This device is described in the September, 1912 issue of the *Bulletin* of the American Institute of Mining Engineers.

The wear of the bucket is confined to the lip and the corners which have to pick up the material. When the lip on the corner of a 7-in. by 12-in. malleable cast-iron bucket is worn down about 3 in., the bucket is of no further value as a carrier and has to be replaced. The worn bucket, containing about three-quarters by weight of the metal of the original bucket, has a
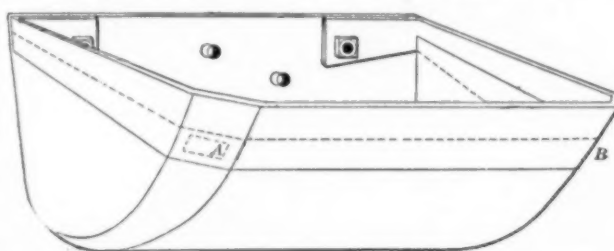

FIG. 1.—ISOMETRIC PROJECTION OF BUCKET WITH LIP ATTACHMENT SHOWING THE LIP BOLTED TO BUCKET.

scrap value of about 10 per cent of that of the new bucket. To the cost of the bucket, however, should be added the time-cost of replacing the old bucket by a new one. This time-cost is considerable when the loss per hour to cease operating the mill, is about $200.

It is out of the question to put lips on a bucket unless the latter was originally fitted with this device. For if it is attempted to fit lips upon buckets nor originally provided with them, it will be necessary to drill holes for rivets and then spend considerable time fitting the lips to the irregular contours of the worn buckets. The improvement suggested is an adjustable attachment which can easily be fastened to, or removed from, the base of a bucket without the use of rivets. This attachment, to be made of a high-grade steel drop-forging or casting, will combine toughness, hardness, and tensile strength. The base of the bucket may be a cheap casting, possibly ordinary cast-iron. This combination of lip and base will not increase

the cost of the improved bucket much, if any, above that of the ordinary bucket now in use, but will increase the life of the bucket, doing the same amount of work, about 400 per cent, with a corresponding decrease in time lost in changing buckets.

This improved bucket is illustrated in Fig. 2, which shows a projection of the bucket with the attachment in place. Two bolts, one on each side of the elevator-belt, pass through the base and attachment and hold both fast to the belt. These end-bolts are 0.25 in. longer than the others. The lip is fitted to the base at the opening at A which receives a lug. There is another opening at B, which is not shown. Probably a more desirable arrangement than this would be to place the attachment inside the base, in which case the outside of the lip and the inside of the base would be rabbeted.

Recent Metallurgical Patents

Iron and Steel.

Briquetting Flue Dust.—A new method of briquetting flue dust has been patented by ROBERT HUBNER, of Elizabeth, N. J. The object of the invention is to produce tough, porous briquets. The mixture is prepared as follows: 60 parts by weight of flue dust, 7.5 parts alumina clay, 84 parts silica, 10 parts lime, 3.5 parts magnesia. These materials are thoroughly mixed, after which there is added 0.6 part hydrofluoric acid, and water in sufficient quantity to convert the mixture into a stiff paste. The mass is then briquetted in any suitable machine.

The percentage of hydrofluoric acid employed depends upon the amount of iron in the flue dust, being increased as the iron increases. Likewise the percentage of silica is dependent upon the amount of silica in the flue dust or in the clay. The addi-
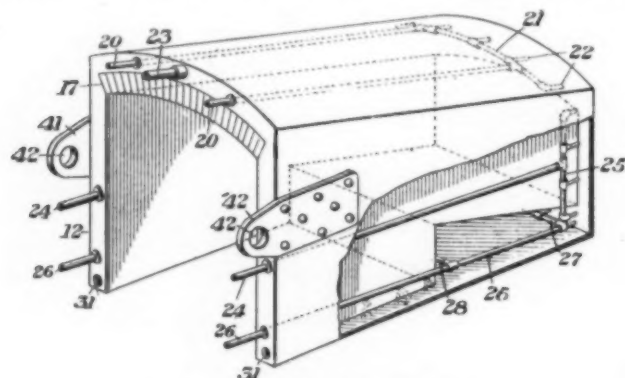

FIG. 1.—WATER-COOLED GAS PORT.

tion of the hydrofluoric acid has the effect of producing a fluid slag from which the iron readily separates. (1,038,370, September 10, 1912.)

Water-Cooled Gas Port.—A novel means for cooling the gas ports of a regenerative furnace is contained in a patent recently granted to LUTHER L. KNOX, of Avalon, Pa., assigned to the Keystone Furnace Construction Company of Pittsburgh, Pa. The water-cooled port is shown in Fig. 1, and its application to the furnace in Fig. 2. The port consists of a hollow metal shell suitably lined with reverberatory material. Inlet pipes are shown at 20, ending in a spray pipe and sprays 21 and 22. Side walls are supplied by pipes 24, also ending in spray pipes at 25. The bottom is provided with an inlet pipe 26, having inner transverse connections 27 and 28. The outlet pipe is shown at 23. Cleaning ports are placed at 31. The entire structure may be readily removed from beneath the arch 9 by means of the side plates 41. (1,038,154, September 10, 1912.)

Gold and Silver.

Cyanide Cylinder.—A proposal to carry on the cyanide process in a cylinder similar to the manner in which the chlorination process was formerly used is contained in a patent granted to HENRY B. MEADE, of Los Angeles, Cal. The in-

ventor has designed a revolving cylinder with an interior filter bottom and a special valve for forcing air, oxygen or solutions into the cylinder under pressure while in motion or at rest.
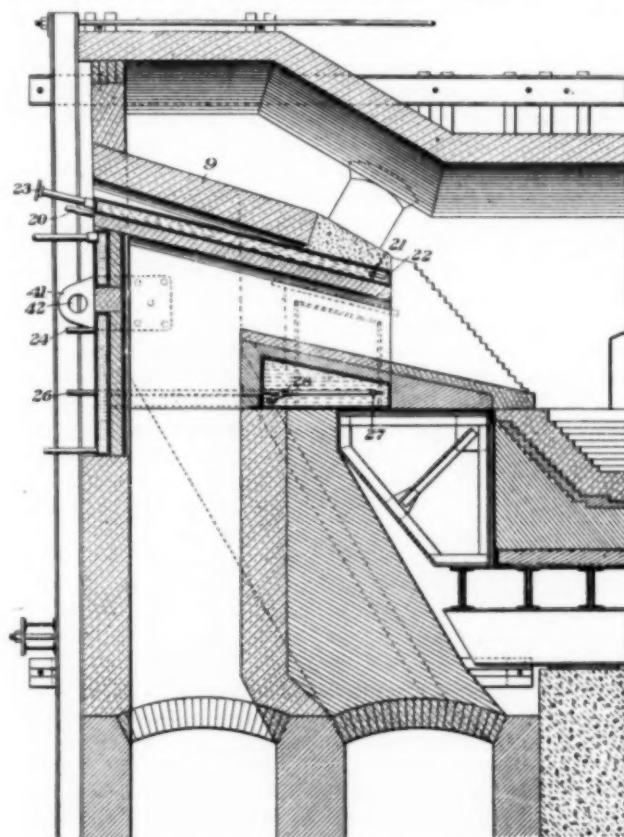

FIG. 2.—COOLED GAS PORT IN REGENERATIVE FURNACE.

Ore is charged and discharged through a manhole, and cyanide solution is withdrawn through a valve below the filter. (1,038, 173, September 10, 1912.)

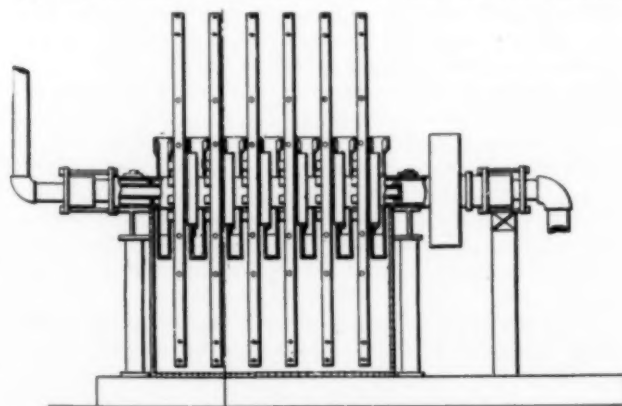**Filter.**—A new application of the leaf form of vacuum

FIG. 3—SECTION OF REVOLVING LEAF FILTER

filter has been patented by CHARLES E. SINGER, of Parral, Mexico. As shown in Figs. 3, 4 and 5, the filter consists of a plurality of leaves vertically mounted on a shaft and suitably connected with a source of pressure and vacuum. Each circular filter is made up of sectors, as shown in Fig. 4. As the shaft revolves the sectors are made to pass through a tank containing the slime which is to be filtered, where a cake is formed in the usual manner. The vacuum is maintained during the revolution of the filter until the sector approaches the discharge launder, where compressed air is ap-

plied and the cake is blown off into the launder. No provision is made for washing the cake, consequently the filtered pulp must be again mixed with solution or water and retreated on a second machine. (1,017,476, February 13, 1912.)

Slime Filter.—Mr. DAVID J. KELLY, of Salt Lake City, Utah, has recently patented certain improvements on his filtering apparatus, consisting in means for operating a series of presses from a central station; in controlling the traveling of the filter carrier by means of a hydraulic brake; and in using
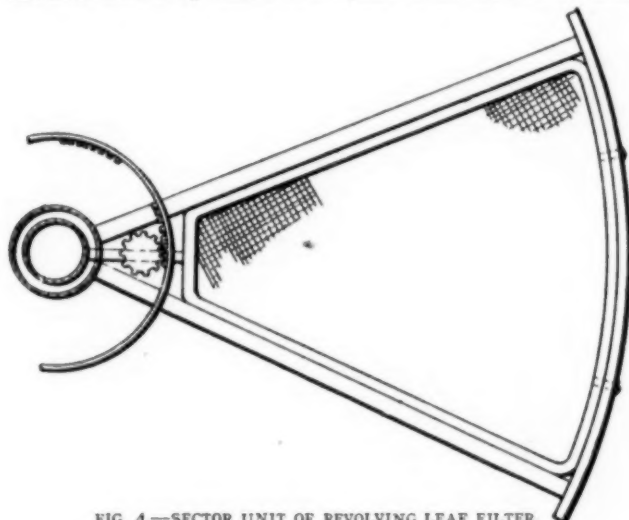

FIG. 4.—SECTOR UNIT OF REVOLVING LEAF FILTER.

a counterweight tank in place of the solid counterweight described in his former patents. (1,015,960, January 30, 1912.)

Copper and Zinc.

Making and Refining Alloys.—The use of boron trioxide as a flux in the manufacture and refining of copper alloys containing tin, zinc, manganese, and so forth, is the basis of a patent granted to WALTER S. ROCKEY and HILLIARY ELDRIDGE,
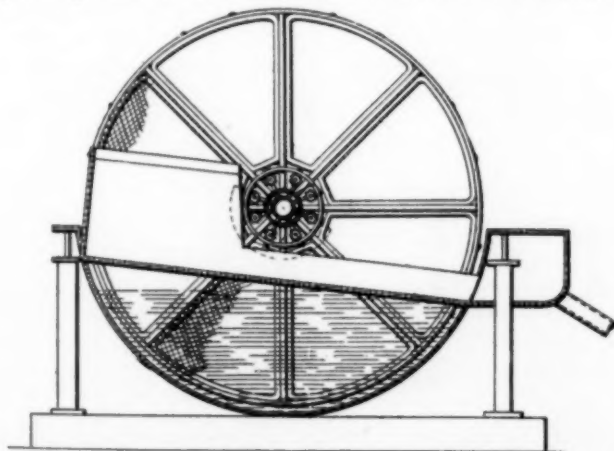

FIG. 5.—SIDE ELEVATION OF REVOLVING FILTER.

of New York City. The method consists in first melting the tin, zinc or manganese under a cover of boron trioxide and then adding molten copper, pouring the latter through the flux. While the copper is being melted before its addition to the alloy metal, carbon is added to assist in the reduction of the oxides that may be contained or formed during the melting. This refining process is further carried on by the boron trioxide flux after the alloy has been made, the molten mixture being maintained at the required temperature until the gases and impurities are expelled. (1,017,629, February 13, 1912.)

Preheater for Zinc Furnaces.—The preheating of the material charged into zinc furnaces is a desirable feature which, however, has not been generally adopted on account

of the difficulties usually encountered. It is the object of a recent invention by CARL ORITZHEIM, of Cologne, Germany, to provide a heater which will overcome the difficulties mentioned. The heater is shown in Fig. 6 applied to a vertical muffle. It is designed to feed the charge through the tubes b, which are of small sectional area and therefore adapted to heat the charge
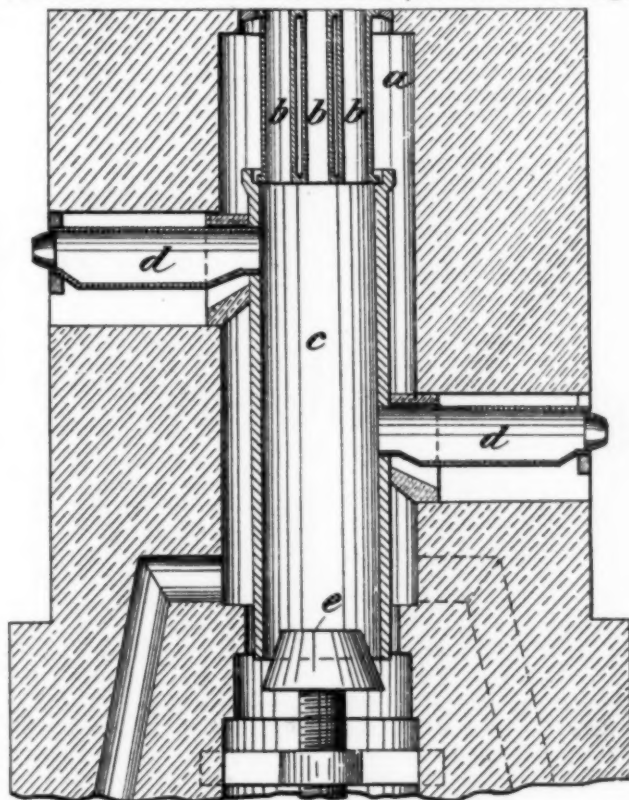

FIG. 6.—CHARGE HEATER FOR ZINC FURNACE.

quickly. From this heater the charge passes into the muffle c below. Zinc vapors are condensed in the receivers d, while the retort residue is removed continuously or intermittently at e. As the charge gradually sinks in the muffle additional ore is fed into the heater. (1,038,205, September 10, 1912.)

Ore Roaster.—A roasting furnace particularly adapted to

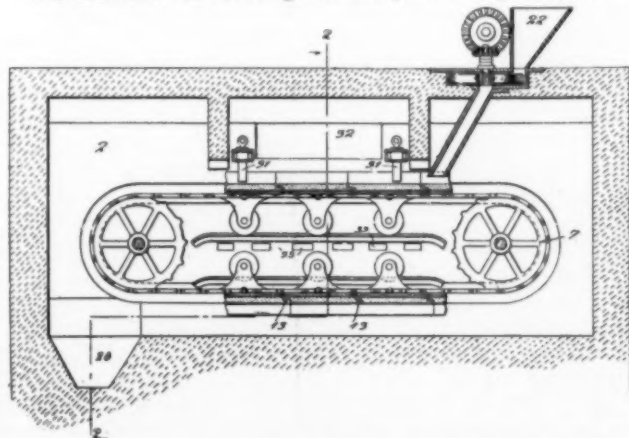FIG. 7.—ORE ROASTER FOR SULPHURIC ACID MAKING.

the roasting of sulphide ores used in the manufacture of sulphuric acid is shown in Figs. 7 and 8, being the invention of CURTIS C. MEIGS, of Woodbury, N. J. As shown in the cuts, the furnace consists of an enclosed chamber, 2, in which is placed an endless sectional hearth, 13, adapted to travel around two sprocket wheels, 7. The furnace is first heated to the

necessary temperature by means of a fire built within the chamber.

The sulphide ore is then supplied from a hopper, 22, on to the traveling hearth. Air is supplied through openings, 33, and the gaseous products of combustion pass out from a flue, 32, to a suitable acid-making plant, as shown. Rabbles are placed at 31 for the purpose of stirring ore and exposing new surfaces to the action of the air. The speed of the hearth is so ar-
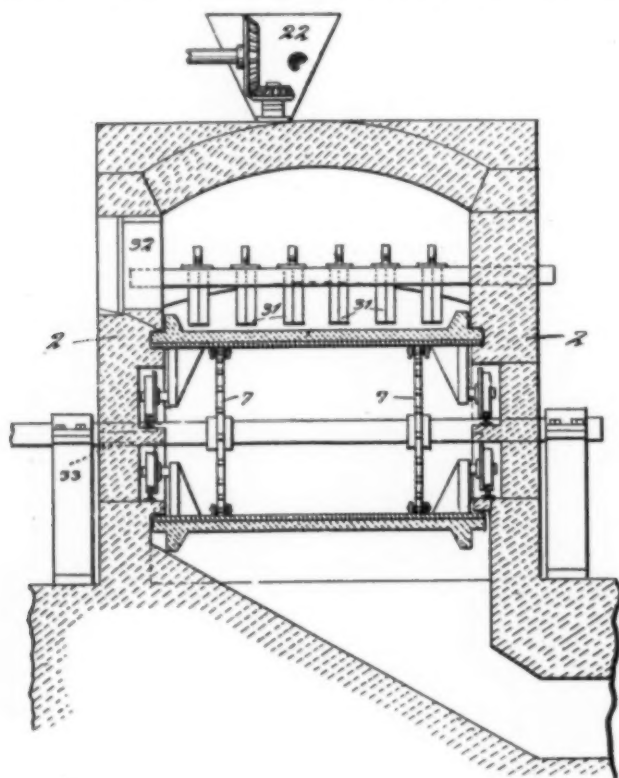

FIG. 8.—SECTION ON LINE 2-2, FIG. 7.

ranged that the roasting operation is completed by the time any section travels from the feed hopper to the point of discharge, where the roasted ore falls into a suitable receptacle at 28. (1,038,408, September 10, 1912.)

Petroleum.

Separation of Oil and Water.—In our issue of June, 1911, page 326, we gave abstracts from patents granted to Dr. Frederick G. Cottrell and associates, of Berkeley, Cal., on electrostatic means for separating water from crude oil. More recently an improvement has been made on the methods formerly described and an additional patent granted to Mr. ALLEN C. WRIGHT, of Berkeley, Cal. Apparatus for carrying out the invention is shown in Fig. 9, and consists of an outer shell, 1, on the inner surface of which are arranged ribs, 4, in a helical course.

The shell then constitues the outer electrode, the inner electrode being the cone 12. The space between the electrodes is filled with the material to be treated, which is fed through the pipe 6.

The outer electrode (the shell) is given a rotary velocity of from 300 to 500 r.p.m. At the same time the material is placed under electrostatic influences, which cause the smallest particles of water to agglomerate. When these particles become of sufficient size to be acted on by centrifugal force they are thrown to the outer shell and forced by the spiral ribs to the bottom of the receptacles, where they are discharged through pipes 9.

The dewatered oil rises and is discharged through pipe 7. (1,034,668, August 6, 1912.)

International Rubber Conference.

The Third International Rubber Conference was held in the new Grand Central Palace, New York, September 24 to 30, 1912. Chemists and engineers, planters and manufacturers and representatives of interests allied with the rubber industry gathered for mutual advantage and benefit. At the meetings which were held, all of the aspects of the rubber business were considered, while the exhibits combined to give tangible evidence of technical progress in an industry which is of growing importance.

Exhibits.

All of the rubber producing countries of the world were represented by wonderful exhibits of wild and plantation rubber
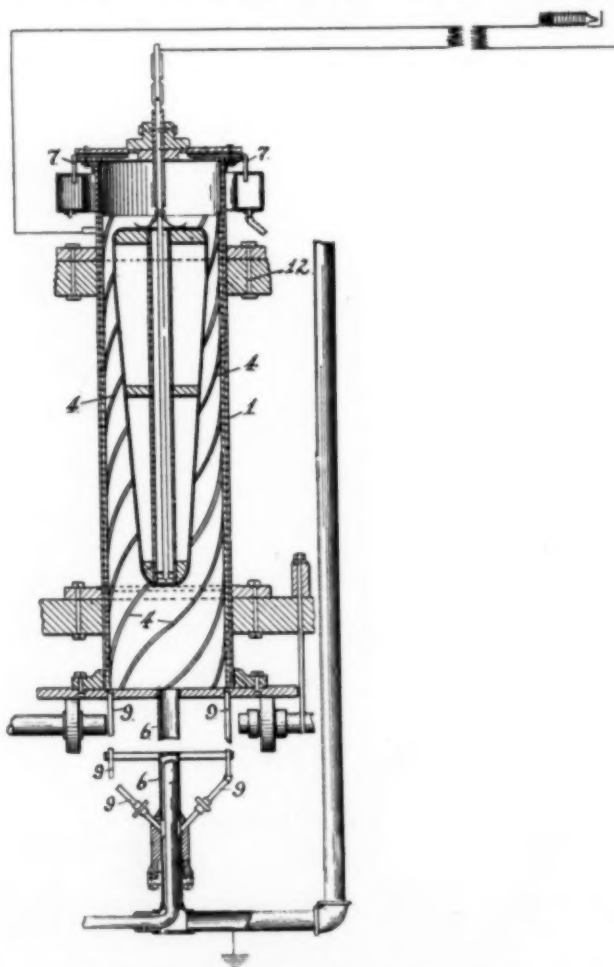

FIG. 9.—ELECTROSTATIC SEPARATION OF OIL AND WATER.

in both the crude and refined state. Brazil easily held first place in point of size and extent of exhibit, showing products from the states of Amazonas, Matto Grosso, Para, Acre, Minas Gereas and Bahia. Other countries having exhibits were, Malay States and Straits Settlements, Ceylon, Hawaiian Islands, Philippine Islands, Burma, Bolivia and Honduras. A cinematograph exhibition at various hours during the day and evening, gave scenes of life and work in the rubber countries.

Of great interest from a technical point of view were the exhibits of companies which make "reclaimed," "regenerated" or "naturized" rubber from old rubber scrap and waste. While the details of many of these processes are kept secret, they consist mainly in disintegrating the old rubber and dissolving the cotton or woolen fabric with acid or alkali, washing and pressing. The product is usually able to compete with the lower grades of primary rubber, like those from Africa. The ten-

sile strength and elasticity of the reclaimed product is less than that of the original.

One of the regenerating systems makes use of an electric current of high amperage and low voltage during the process of dissolving the fabric. The claim is made that the time required for disintegration and solution is reduced one-half by the use of the current—say from 22 hours to from 8 to 11 on various grades of scrap.

A number of interesting exhibits were prepared by manufacturers of rubber-making machinery and of chemical products used in compounding rubber. The New Jersey Zinc Company had on exhibition its zinc ore from Franklin furnace, and the zinc oxide product which is used as an admixture in rubber articles where toughness is required. Aluminium flake is another trade product used for this purpose. Several producers of natural bitumen and mineral wax also displayed their products, as well as admixtures of these and rubber gum. The effect of these substances is greatly to increase the strength of rubber and to prolong its life. The fertilizer business was represented by the German Potash Syndicate.

Werner & Pfleiderer Company, Saginaw, Mich., showed their apparatus for masticating and washing rubber.

Buffalo Foundry & Machine Company, Buffalo, N. Y., exhibited a vacuum drying apparatus for drying rubber, and also showed their facilities for turning out special iron castings.

C. J. Tagleabue Manufacturing Company, of Brooklyn, N. Y., displayed thermometric and vacuum recorders, in connection with other mechanical exhibits.

J. P. Devine Company, of Buffalo, N. Y., had in operation a commercial vacuum dryer, pump and condenser for the drying of sheet rubber.

Meetings and Papers.

Daily sessions were held for the discussion of chemical and engineering problems connected with the rubber industry. Great interest centered in the subject of standard methods of analysis and standard physical tests. In the matter of specifications for rubber goods, the United States and foreign countries, and several scientific societies were represented by official delegates who hoped to standardize this branch of the industry. Among the other subjects discussed were the possible rubber producers in the temperate zone, and the commercial possibilities of synthetic rubber.

Problems in Vacuum Drying.

The merits of the vacuum drier as a means of removing the last traces of moisture from rubber, were presented by Mr. J. P. Devine, of Buffalo, N. Y. From his paper we make the following excerpt:

"My introduction to and connection with the rubber industry was made about ten years ago when I sought to interest manufacturers in this country in a new method of drying rubber. I found, however, that the manufacturer was disposed to be quite satisfied with the then existing method. * * *

"Notwithstanding the discouragements of my early efforts, I made investigation as to methods of drying rubber and learned that little, if any, advancement or improvement had been made up to that time, and this confirmed my opinion of the value of the vacuum process, and I have maintained ever since its introduction that the vacuum drying process and apparatus introduced in Europe by Messrs. Emil Passburg and William Strohn, and in this country by myself, would be and has since proven of inestimable benefit to the rubber industry.

Old Methods and Their Disadvantages.

"For years rubber was dried by any convenient means that would not represent an expenditure of capital, the boiler room, or a room adjacent thereto, serving the purpose until the effects of the elements upon the rubber began to receive attention and study. And then, the improvements consisted largely in the construction of drying lofts, the racks on which the sheeted rubber was hung and the distribution of heat by coils, or by circulation of hot air by means of fans and blowers; little regard however being given to temperature or length of time of the drying process. Until the introduction of the vacuum drying apparatus this primitive method was still in use and occasionally an advocate is still found, who asserts that the hot air method is necessary for the proper curing of some particular grade of rubber. The fallacy of such assertions is proved by the use of the vacuum apparatus drying every grade of crude rubber, and I shall endeavor to make clear to you the claims I make for the superior quality of rubber dried by a properly designed and constructed vacuum apparatus.

"While it is true, considerable thought was given to improve processes for drying rubber, there were no striking departures from the antiquated method of using hot air as the heating medium. The construction of special drying rooms from which direct sunlight was excluded; provisions to eliminate dust and dirt; the regulation of temperatures for various grades of rubber, as well as the attempt to dry the air before being admitted into the drying room, all contributed to avoid the deterioration of the rubber by such means; but the value of these improvements was doubtful as they only tended to reduce the effect of high temperatures with a consequent prolongation of the drying period.

"The fact is that the two insidious enemies of rubber are heat and oxygen and these elements are, and always will be present, and necessarily so, in any system of hot-air drying. They are deteriorating agents and their elimination is most essential for the proper drying of rubber. Their elimination by the vacuum apparatus has proven the superiority of the vacuum-dried rubber in the processes of its manufacture.

"Another and serious objection to the hot-air system of drying rubber is, that rubber as it comes from the washing machine, contains a very large proportion of mechanically bound moisture; while this is readily given off in the hot-air drying room, its expulsion causes a contraction of the rubber, which, with the oxidation constantly taking place, causes a hardening of the surface that prevents the elimination of the last moisture within the rubber, except by a very prolonged drying period, during which time the rubber is further subjected to oxidation and not unlikely to excessive heat. Unless the last traces of moisture are eliminated, 'blowing' is sure to result during the following stages of its manufacture. * * *

Advantages of Vacuum Drying.

"The deteriorating agents—oxygen and excessive heat—can only be eliminated by the vacuum process and apparatus, which afford the proper conditions to dry rubber rapidly, uniformly and thoroughly at a low temperature and without oxidation, independent of climatic conditions.

"It must be borne in mind that under atmospheric conditions a rapid boiling can only take place at 100 deg. C. or 212 deg. F. and that as the temperature decreases, the drying time is extended; while under vacuum, the boiling point is greatly decreased. To illustrate, under a vacuum of 29 in. water boils at 25 deg. C. or 77 deg. F. Rubber dried in the vacuum chamber, while the first free water is being removed, will not be heated practically above the boiling point of water at that particular vacuum; as the moisture is evaporated from the rubber, naturally the temperature of the rubber being dried tends to increase; to prevent any over-heating, the supply of the heating medium—steam or hot water—is regulated accordingly and entirely shut off before the final drying; the last traces of moisture are therefore drawn off by the latent heat in the dryer accelerated by the high vacuum. Because seemingly high temperatures are used at the beginning of the drying process to expedite evaporation, the erroneous impression is sometimes formed that the rubber is overheated in the vacuum chamber; but it must be obvious that in a properly constructed vacuum chamber with its auxiliaries—condenser and pump—properly

balanced, the application of well-known physical laws absolutely prevent any over-heating, if reasonable care is only taken in its operation.

"Then too, the drying process taking place under a high vacuum in the absence of oxygen, oxidation cannot take place. The rubber not being overheated nor impaired by oxidation has greater elasticity and tensile strength, and accomplishes that which is the aim of every manufacturer, to obtain the greatest yield when being made into final product.

"As different grades of rubber must be subjected to varying temperatures, the temperature of the heating medium is easily regulated without over-heating or materially affecting the drying time so that the vacuum process offers many advantages over the old method of drying rubber, whether it be fine Para or Pontianac.

"Another very great advantage of the vacuum process lies in the fact that as the drying period is only a few hours, varying according to grade of rubber, the crude rubber can be washed, dried and processed in a fraction of a day; consequently there can be no deterioration after drying and before using, as it is well known that washed rubber oxidizes very rapidly, and there is no danger of the final manufactured product being porous or spongy due to the presence of moisture. * * *

Financial and Other Economies.

"In an average establishment of today, making a general line of rubber goods, two tons of crude rubber is a conservative estimate of its consumption. If the old hot-air method is used, in order to properly and thoroughly dry the washed and sheeted rubber, six weeks are consumed in the drying process. Seventy-two tons of rubber would be hanging in the drying lofts, which at $1.00 per pound, would represent an idle investment of $144,000 on raw material, the carrying charge alone at 5 per cent amounting to upwards of $20 per day; and should the carrying charges for insurance, factory space, etc., be included, the above sum would be greatly increased.

"The same quantity of rubber could be more thoroughly and permanently dried by one or two vacuum chambers in a day of ten hours, so as to "work up" whatever grade may be required for each day's output and the initial cost of such an installation would be less than the cost of the old-fashioned drying rooms for the same quantity. So that the vacuum chamber pays for itself in the savings on investment, carrying, insurance and other fixed charges on raw material, as well as giving a flexibility to the factory for its daily production that cannot be obtained by any hot-air method.

"To illustrate the great saving in factory space, a vacuum drying chamber having a capacity of approximately two tons of dry sheeted rubber per 10 hours, occupies a space of 8½ ft. high, 15 ft. wide and 9 ft. long, and its auxiliaries, the condenser and the pump, can be conveniently located at any place in the factory in proximity to the dryer.

"In addition, the vacuum drying process offers further advantages over the hot-air system because of the other factory economies derived by its use. The expense for a hot-air system for steam consumption and attendance is enormous. It is an established fact that by any hot-air system not more than one-third of the heat units supplied are utilized in the evaporation of moisture, whereas by the vacuum process practically every heat unit is transmitted to and comes in direct contact with the material being dried. The saving of fuel is consequently most considerable not only on account of shortening the drying time from weeks to hours, but because of the much higher efficiency of the heating medium used in the vacuum drying process.

"The vacuum drying apparatus is constructed of cast iron and is practically indestructible; there are no appreciable maintenance charges and fire hazards are eliminated.

"As to attendance, only one operator is required to operate the vacuum dryer, and while one charge is being dried, he is engaged in filling the second set of trays with rubber for the next charge; and in recharging the apparatus, replaces each tray of dried rubber with one of wet.

Design and Construction of Apparatus.

"A vacuum drying apparatus consists of a vacuum drying chamber which should be constructed of a special grade of close grain homogeneous cast iron equipped with heating shelves made of hydraulically straightened sheet steel plates; I emphasize that the plates should be hydraulically straightened as by this means only can a uniform even and flat surface be secured on which the trays are placed and which will withstand the strain of the varying steam pressures used during the drying process; it is important that the heating shelves are so constructed as to insure an even surface as any distortion of the heating shelves will cause the trays to rest unequal and thereby permit an uneven transmission of the heat to the material being dried.

"Connected with the vacuum chamber is a condenser and it is most important that this auxiliary be of proper condensing capacity for the vapors given out during the drying process. * * * The size and capacity of the dry vacuum pump is of equal importance to the well balancing of a vacuum apparatus, for in order to secure the highest efficiency it is necessary to create the highest obtainable vacuum as quickly as possible and when obtained to maintain a uniform vacuum throughout the drying period.

"Complaints have occasionally been made by some people who have attempted to dry rubber under vacuum, that such vacuum-dried rubber was not satisfactory, and the blame was promptly put on the vacuum drying apparatus and process.

"I may state that though these complaints were, of course, bona-fide, the cause for such complaints was misplaced. The vacuum apparatus and process have the advantage of working almost automatically and require scarcely any attendance, but these advantages have in the above-mentioned cases been abused and of course not without detrimental effects to the vacuum-dried rubber.

"To illustrate what I mean, a certain size of a vacuum apparatus is intended to receive a certain quantity of rubber per charge and naturally the heating surface of the dryer, which is equivalent to its drying capacity, has, as I have stated, to be most carefully calculated for this purpose; and if the instructions for operating such apparatus, which are most simple, will only be followed, I might say it is a physical certainty that the vacuum-dried rubber will be satisfactory and far superior to air-dried rubber because with a vacuum apparatus and process nothing is left to chance or climatic conditions.

"In cases, however, where complaints have arisen, the well-meaning people who were using such a dryer, being surprised at the capacity of the apparatus far exceeding their expectations, thought it right to go a little further by increasing the charge of rubber and ultimately loading the apparatus with a much larger quantity than was intended. Of course, it was soon found out that the increased charge could not be dried in the stated time, nor with the stated temperature of heating steam. As it is only human not to decrease one's desires, the natural human remedy was resorted to, that is, an increased temperature of heating steam and also a prolonged drying time. If you consider that the heating surface at a certain temperature within the dryer is intended for a layer of rubber of a uniform and certain thickness, its capacity, or rather the beneficial results obtained therefrom, will be destroyed or at least impaired by an increased quantity of rubber per charge and an increased temperature of heating steam, because the heating surface itself remains the same and it is this factor which remains constant that upsets the results sought to be obtained by the violation of well known, but not considered, natural laws.

"Experience has taught us to balance the necessary heating surface, to transmit a certain temperature to a certain layer

of material to be dried; and it is a quite erroneous, though common assumption, that the same beneficial results may be obtained from a larger quantity of material by simply increasing the thickness of the drying material and increasing temperature in the belief that the above mentioned factor would increase proportionately. This, however, is not the case, as I will more fully point out.

"If one takes the conductivity of rubber alone into consideration, and the gradual but decreasing evaporation of the water contained therein, it can very easily be understood that by altering some of the factors the physical laws on which our calculations are based will be violated without intention, and the penalty is an unsatisfactorily dried rubber—the cause of which is naturally placed at the wrong door. The fault is not in the apparatus, but in the method of its operation.

"The same remarks refer to the auxiliaries of an apparatus for drying rubber and these auxiliaries consist of a condenser and vacuum pump which are both calculated to correspond with the capacity of the vacuum apparatus they are intended to serve.

"To illustrate what I mean: A vacuum dryer of a certain drying capacity and calculated for a certain purpose is intended to evaporate a certain quantity of water in a given time and of course what is essential, at as high a vacuum as is possible under practical working conditions. All this, is to a great extent, based on practical experience with the very material our apparatus is used for.

"If, however, the condenser, instead of handling the quantity of vapor for which its cooling capacity is calculated, is burdened with ever so much larger a quantity, the result must be detrimental in two ways; it not only reacts on the dryer and the product it is supposed to turn out regardless of time, but also reacts on the working of the pump.

"As regards the vacuum dryer, it is essential to have its inner space continuously freed from the vapor arising from the drying material in order that no inner pressure may be created in such apparatus to lower the vacuum. This can only be done by having the arising vapors taken care of in their entirety during their passage through the condenser, the capacity of which cannot be changed at will.

"If more vapors are created than the condenser is intended for, such vapors will partly remain in the dryer (and create inner pressure); the inner pressure thus created consequently reduces the vacuum in the dryer and in further consequence the boiling point of the water contained in the rubber increases and the rubber will be heated up to a temperature never intended, with detrimental effects to the quality of the rubber. The overcharging, as I said before, affects the efficiency of the pump and prevents it from creating the desired high vacuum. The reason for this is that a dry vacuum pump—the only type in question we have in mind while talking on this subject—is intended, dimensioned and constructed for pumping air and not vapor, particularly as the latter expands so enormously under vacuum. If the pump was intended to exhaust rarified or expanded vapor in addition to rarified or expanded air, its dimensions would be so enormous as to make its use practically impossible.

"If, therefore, the dry vacuum pump has to exhaust vapors which have passed uncondensed through the over-taxed condenser, a burden is placed on the pump for which it was never intended; its work becomes inefficient and most naturally impairs the vacuum and efficiency of the whole installation for drying purposes.

"The use of the vacuum drying process was, of course, first introduced and adopted by the manufacturers to dry washed rubber. In recent years, however, much greater thought and study have been given to the preparation of the crude rubber on the plantation; especially is this noticeable in the quality of the rubber reaching the market from the Malaysian, Straits Settlements, and Congo plantations, where vacuum drying apparatus has been installed and found to be of great value.

"This improved quality of the raw material will not cause the discontinuance of the drying process in the factory. It will, however, tend to restrict the operation for the treatment of such rubber to remove the final percentage of moisture only, and thus simplify factory operation."

Standard Methods of Sampling for Check Analysis.

The Association of American Steel Manufacturers has just adopted a set of rules to be followed in obtaining samples for the check analysis of steel. This latest activity of the association is in response to the need that all manufacturers and many consumers have felt for uniform conditions which should govern both parties when steel is to be analyzed, especially when its *acceptance* or *rejection* depends upon check analyses.

The booklet is entitled "Standard Methods of Sampling for Check Analysis," and it is issued under the new classification of "Manufacturers' Standard Practice" which the Association now uses for certain of its adopted rules, when they are not strictly of the nature of specifications.

In the introduction of the pamphlet it is pointed out to be a recognized fact that the different parts of a piece of steel are likely to vary in composition. This variation occurs principally between the center and the outside, and to a slighter extent is dependent upon the position of the piece in the ingot, and the size of the ingot.

Where a sufficient number of check analyses have been made from drillings properly taken at different points in the heat to represent it fairly, their average has been found to compare favorably with the ladle analysis, which is the analysis of a small test ingot taken at any time during the pouring of the heat.

From this it is evident:

1. That the ladle analysis is more representative of the composition than any single analysis of the finished material.

2. That drillings for check analysis, to be fairly representative, should be taken at a point intermediate between the outside and the center of the cross-section.

3. That a sufficient number of check analyses of different pieces should be made to afford a fair average to compare with the ladle analysis.

1. Points to Be Observed in the Sampling of Material for Check Analysis.

(a) Each heat in a lot shall be considered separately, and pieces for sampling shall be taken to represent the heat as fairly as possible.

(b) Samples must be drillings or chips cut by some machine tool without the application of water, oil or other lubricant, and shall be free from scale, grease, dirt or other foreign substance. If samples are taken by drilling, the size of the drill shall be not less than $\frac{1}{2}$ in. or more than $\frac{3}{4}$ in. in diameter.

(c) Samples must be uniformly fine and each must be carefully mixed before analysis.

(d) In referring samples to the manufacturer or other analysts for check analysis, a piece of the full-size section, when possible, should be submitted rather than cuttings, unless the latter are specially requested.

(e) Where material has been subjected to heat treatment other than annealing or simple cooling, subsequent to its manufacture, it should be annealed before sampling.

(f) Check analyses are not representative of the original material when its composition has been altered in any way by some operation such as casehardening, overheating, etc.

2. Methods of Obtaining Samples for Check Analysis

Material has been divided into the following classes, depending upon the manner of sampling:

I. MATERIAL SUBJECT TO PHYSICAL REQUIREMENTS.

Samples for check analysis shall be taken from a test specimen. Where it is required to make additional check analyses, samples shall be taken as indicated under II.

II. MATERIAL NOT SUBJECT TO PHYSICAL REQUIREMENTS.

(a) *Special cast, rolled or forged, semi-finished or finished material of large size, such as ingots, blooms, billets, slabs, rounds, shapes, etc., subject to acceptance on check analysis.*

Samples shall be taken at any point midway between the outside and the center by drilling parallel to the axis. In cases where this method is not practicable, a piece may be drilled on the side, but drillings shall not be taken until they represent the portion midway between the outside and the center. See Fig. I.

(b) *Small or thin material, such as plates, shapes, bars, etc., subject to acceptance on check analysis.*

Material for which the previous method is not applicable shall have samples for analysis taken entirely through the material
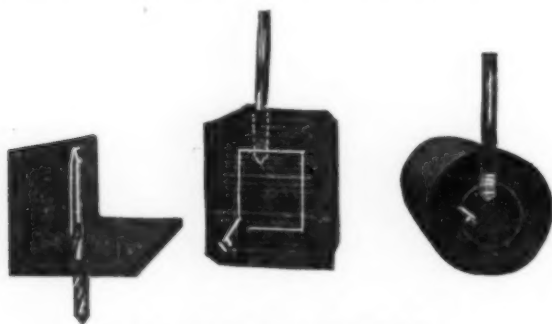
METHODS OF DRILLING SAMPLES

at a point midway between the outside and the center, or by machining off the entire cross-section.

(c) *Commercial material subject to acceptance on ladle analysis.*

The methods described under II (a) and (b) shall apply, except that samples shall be taken at any point one-third of the distance from the outside to the center.

3. Methods of Analysis.

Analyses shall be made by well-known accurate methods. Carbon shall be determined by the combustion method.

4. Rejection of Material on Check Analysis.

Any rejection of material ordered to a specific chemical range shall be based on the following:

(a) The minimum number of samples to be taken from a heat before rejection by the purchaser shall be as follows:

Weight in Gross Tons.	Minimum Number of Samples.
5 or less	3
10 or less but over 5	4
15 or less but over 10	5
Over 15	6

In case the number of pieces in a heat is less than the number of samples given, one sample from each piece shall be considered sufficient.

(b) Separate determinations shall be made on each sample and the results averaged, unless they clearly indicate mixed grades.

The Association of American Steel Manufacturers comprises thirty-two steel manufacturing companies. Mr. A. A. Stevenson is the president; Mr. Jesse J. Shuman, care of Jones & Laughlin Steel Company, Pittsburgh, Pa., is the secretary, from whom copies of the different publications of the Association on Standard Specifications and Standard Practice may be obtained.

The Ohio Copper Company, Bingham, Utah, has issued a statement of operations covering the period from October, 1911 to March, 1912. The copper content of the crude ore was 1.176 per cent, and of the concentrate 22.179 per cent. The ratio of concentration was 33.74:1, and the quantity treated was 311,067 tons. The loss in concentration was 44.13 per cent.

Convention of Foundrymen's Association and Institute of Metals.

The allied foundry interests of the United States convened in annual session in Buffalo, N. Y., September 24 to 27, 1912, to consider the past year's progress in the metal founding industry. The ever growing importance of this branch of metallurgy was again indicated by the very large attendance and the number of interesting and important papers presented.

Three different organizations were in session: The American Foundrymen's Association, The American Institute of Metals, and The Associated Foundry Foremen. All joined in an opening meeting on September 24th, presided over by Mr. Joseph T. Speer, president of the American Foundrymen's Association. Addresses of welcome were made by Mayor Fuhrman, of Buffalo, and by the president of the Chamber of Commerce. Response on behalf of the visitors was made by Mr. A. E. Howell, of Nashville, Tenn., one of the vice-presidents of the Association.

Entertainment

On Tuesday morning a visit to the Roycroft Shops, with dinner at the Roycroft Inn, was arranged for the ladies. In the evening the Associated Foundry Foremen held their annual banquet while the ladies attended a theater party. A boat ride in the harbor and down Niagara river was given on Wednesday morning.

The entertainment of Thursday comprised a sight-seeing automobile tour of Buffalo, in the morning, and a subscription dinner at Hotel Statler in the evening. Secretary Nagel, of the Department of Commerce and Labor, and Elbert Hubbard were among the speakers at the banquet.

The annual display of the Foundry and Machine Exhibition Company was open daily to visitors at the Broadway Arsenal, Buffalo's new exposition building. It was again a brilliant success.

In the following we give abstracts of two papers presented. Further abstracts are reserved for our next issue.

The Electric Furnace for Brass Melting.

A paper on this subject was presented by **Mr. G. H. Clamer** and **Dr. Carl Hering**, of Philadelphia, Pa.

The paper summarizes the advantages of the electric over the fuel furnaces for brass melting and points out the incorrectness of the prevailing belief that owing to the necessarily greater cost of electric over fuel heat, it would not pay to use electric heat for such a low melting alloy as brass. After giving a table of the cost of a kilowatt-hour of heat from electric energy, water power, wood, coke, oil and coal, assuming perfect conversion, the authors show that although the discrepancy between this cost of electric and fuel heat is great, there are many other factors of money value which more than balance this difference.

One is the far greater efficiency, that of the oil furnace being only about 10 per cent and that of the electric from 65 to 75 per cent. In fact it would take less oil to operate an oil engine, generate electric energy with it and use this in a furnace than to melt the metal directly with oil. Moreover there is no oxidation, no combination with sulphur from coke gases, less volatilization of zinc, sharper and less faulty castings, less trouble and loss in running together finely divided scrap metal. It is shown that the saving of metal alone, amounting to probably about 20 cents per 100 lb., more than pays for the electric energy, besides saving the cost of the oil or coal and the crucibles. The authors favor the combination of preheating with fuel heat and supplying only the higher temperature heat electrically.

A table gives the kw-hours required for melting and superheating 100 lb. of various metals and alloys, assuming no losses.

The various types of electric furnaces available for brass are discussed briefly and their advantages and disadvantages pointed

out, more particularly the resistance furnace with which the authors have experimented and which they are developing. In this (the so-called "pinch effect furnace") the heat is generated in a small portion of the liquid metal which is then immediately forced out into the bath in the hearth; cold metal is melted by immersion. The heat is generated in the metal itself, where it is wanted, hence at high efficiency; it can be forced to a high degree whereby the size of furnace for a given output is increased. The systematic forced upward circulation brings all suspended impurities to the top, hence purifies the metal. Being a closed form of hearth the loss of zinc will be very slight. A moderately large furnace of this type for bronze melting is under construction and is expected to be in commercial operation very soon.

The authors are of the opinion that melting brass in a suitable electric furnace is not only not more expensive, but will actually be cheaper when all factors, including the smaller loss of metal, are taken into consideration.

Electric Melting for Copper and Brass.

Another interesting paper on the use of the electric furnace in the copper and brass industry was presented by Mr. **C. A. Hansen,** of the General Electric Company, Schenectady, N. Y. In the first part of the paper figures are given on the performance of three commercial oil furnaces, which he then compares with the use of the electric furnace.

Experiments on the electric melting of copper were made in a furnace, designed by Mr. C. A. Weeks, of Philadelphia, Pa., primarily for the distillation of zinc. From the results of the tests the author figures the power required to melt copper after the furnace had been warmed up. Two different methods of reasoning lead to 298 and 243 kw-hours per ton of copper respectively corresponding to efficiencies of 65 and 80 per cent.

The oil burned in an oil-fired furnace would, if burned in a modern oil engine, supply more than enough electrical energy to melt the same amount of metal in the electric furnace.

With respect to fuel and energy cost the following comparison is made: With oil at 2.5 cents per gallon, power on a 3-heat, 12,000-lb. per day output will have to sell between 0.40 cents and 0.82 cents per kw-hour to equal actual costs of oil and blast in oil-fired furnaces where energy to the blower is supplied at the very nominal rate of 1 cent per kw-hour and where 1.78 to 3.46 gal. of oil are used in melting 100 lb. of averaged red brass.

Even the lower figure, 0.40 cent per kw-hour, is guaranteed by some of the oil engine concerns selling engines of the capacity necessary to operate a 2-ton furnace, contingent on 2.5 cents per gallon.

The higher figure, 0.82 cent per kw-hour, is obtainable for day loads from a great many of the central stations in the various cities in this country. The electric melting of brass and copper is, therefore, on a more attractive basis as regards energy costs than is the electric melting of steel and the latter undoubtedly has a wide commercial application for castings.

As to the loss of metal "it is certain that if equally careful crews operate electric furnaces and oil-fired furnaces the former will have a decided advantage."

Theoretically, the electric furnace has an immense advantage over the ordinary oil-fired furnace, in that the electric furnace may be practically sealed to prevent access of air—which means that so long as the vapor pressure of the zinc in the charge does not exceed atmospheric pressure the maximum possible loss of zinc must be the furnace full of zinc vapor.

The author finally compares briefly the relative advantages and disadvantages of the induction furnace, the direct arc furnace, the indirect arc furnace, and the resistance furnace for the special purpose of brass melting.

On account of limitations of space, further notes on the two papers, abstracted above, as well as on quite a number of other interesting papers must be reserved for our next issue.

Non-Ferrous Metal Market.

With the exception of copper, the price of all the principal non-ferrous metals has appreciated materially during the last month. There has been a decided advance in lead and a strong upward tendency in tin and spelter. The domestic tin market has not kept pace with the bull speculation in Europe, but it is expected that prices may go still higher. Consumption of all the metals is fairly good.

Copper.—There has been no special feature in this market. Electrolytic has sold freely and Lake has been in moderate demand, but sellers show no anxiety to push sales. The last New York quotations are: 17⅝@17¾ cents for Lake and 17.45 @17.50 cents for electrolytic.

Tin.—This market has reflected European operations and has shown great strength, reaching nearly 50 cents on September 16. The latest reports give a price of 48¾ cents for September delivery.

Lead.—On Sept. 9 the principal producer raised the price to 5.10 cents and demand has continued good. The metal is regarded as holding a strong position. Price at New York is 5.10 cents, and at St. Louis, 4.95@5 cents.

Spelter.—A fair business has been done at advancing prices, but importations have already been made which have had the effect of checking a further rise and placing New York quotations below their parity with St. Louis. The last quotations are 7.45@7.55 cents New York, and 7.35@7.45 cents, St. Louis.

Other Metals.—Business in aluminium continues good with prices firm at 24¾@25 cents, New York. The antimony market has improved and quotations range from 8⅛ to 8¾ cents for various grades. Quicksilver is steady at $42 per flask of 75 lb. New York, and $41.50 San Francisco.

Personal.

Mr. **James F. Callbreath,** secretary of the American Mining Congress, is at Spokane completing arrangements for the annual convention which will be held November 25 to 30.

Dr. **Joseph A. Holmes,** director of the Bureau of Mines, will be one of the principal speakers at the American Mining Congress in Spokane, November 25 to 30.

Mr. **George F. Comstock,** metallographist with the Park works of the Crucible Steel Co., Pittsburgh, has resigned to take charge of the new physical testing laboratory recently erected at a cost of $10,000 by the Titanium Alloy Mfg. Co., at Niagara Falls, N. Y.

Mr. **Charles Vickers,** editor of the non-ferrous casting department of the Foundry, has resigned to take charge of introduction of copper-titanium for the Titanium Alloy Mfg. Co., Niagara Falls, N. Y.

Mr. **B. Magnus,** general superintendent of the Electrolytic Smelting & Refining Co., Port Kemble, Australia, has been appointed general manager for the Mt. Morgan Gold Mining Co., of Queensland. He will continue to act as consulting engineer for the refining company, which treats all the blister copper from the Mt. Lyell and Mt. Morgan mines.

Mr. **A. W. Cole** has taken a position as mill superintendent with the Concheno Mining Co., at Temosachic, Chihuahua, Mexico.

Mr. **A. F. Flynt,** metallurgist, of Tepic, Mexico, is traveling in the United States.

Prof. **R. C. Benner,** of the Department of Industrial Chemistry in the University of Pittsburgh, spent part of the summer vacation in Ontario.

Mr. **R. B. Watson,** manager for the Nipissing Mines, of Cobalt, Ontario, Canada, is in Germany taking baths for the relief of rheumatism. During his absence Mr. **Hugh Park** is in charge of operations.

Mr. Bernard McDonald has returned to Los Angeles from a trip to the Cobalt district. He is making his home temporarily in Los Angeles during the Mexican troubles.

Mr. E. M. Hamilton is at Cobalt, engaged in completing the new Nipissing mill which will shortly be put in commission.

Mr. Walter Douglas, who was reported captured by Mexican rebels in Sonora, is now known to have escaped to Agua Prieta, using a railroad motor velocipede.

Mr. Paul W. Webster has resigned his position with the M. H. Treadwell interests, and after October 1 will devote his time to the business of the Lutz-Webster Engineering Co., which he has formed with Mr. William H. Lutz. The new company is an incorporation of the Grays Ferry Machine Co., Philadelphia, and will continue the same line of work at the same address, 3103 Grays Ferry Road.

Obituary.

Mr. T. Lane Carter died on September 3, in Montana, as the result of injuries received while examining the Polaris mine. Mr. Carter entered active engineering in South Africa, where he was on the staff of the Crown Deep and Robinson Central Deep mines. He had spent several years in foreign lands, engaged in the practice of his profession, and was a contributor of timely articles to the technical press. Mr. Carter was married just a few months prior to his death.

Mr. S. C. Hazleton, superintendent of the Midvale smelter of the United States Smelting Co., Utah, was killed in an automobile accident near Salt Lake City on the evening of September 6. The car in which Mr. Hazleton was riding was struck by a street car while trying to avoid a collision with a carriage. Mr. Hazleton was about 35 years of age and unmarried. Coming to Utah a few years ago, he first took charge of the Murray plant of the American Smelting & Refining Co., going later to his present position.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

597,820, Jan. 25, 1898, Nathaniel S. Keith, of Chester, England.

Relates to obtaining gold and silver from ores or materials containing the same. The apparatus consists of an upper tank containing ore to be treated, an intermediate electrolytic cell and the lower tank to receive the solution from the cell. The ore is treated with a solution, which is then electrolyzed and then run off and pumped back onto the ore again. The process is as follows: To a solution of cyanide of potassium varying from 0.01 per cent to 5.0 per cent in strength, a solution of mercury is added in a solution of cyanid of potassium, for instance oxid of mercury in a solution of cyanid of potassium. The solution of mercury is added preferably in quantity sufficient to supply an ounce of mercury to each ounce of gold to be dissolved. Having prepared the solution, the gold- or silver-bearing materials are submitted to its action in the same manner as practiced in vats or tanks. In a solution of cyanid of potassium, gold is electropositive to mercury and is capable of combination with the equivalent acid constituent of mercury associated with the solution of cyanid of potassium. The gold thus becomes much more freely soluble in the cyanid of potassium solution. It is necessary, in order to insure speedy dissolving of gold by a solution of cyanid of potassium, to have the particles of gold in contact with some material electronegative to it in that solution. Besides mercury, iron and carbon both possess this quality. In the case of iron, it is used in solution, as just described when mercury is used. If carbon be used, it is mixed in a pulverized or other finely-divided state with rocks or ores likewise finely pulverized or divided so that the particles of carbon may be more rapidly brought into contact with the particles of gold or silver. When the solution is drawn from the vats it is passed through an electrolytic cell continuously, so that the solution entering the vat runs out at another place, exhausted of its gold and silver constituents, into another vat containing gold- or silver-bearing materials. For the anode, zinc or iron in the form of plates is used. The surfaces of the zinc are preferably amalgamated. Mercury may also be used as an anode. Two kinds of electrolytes are used, one for the anode and one for the cathode. When zinc is the anode a half saturated solution of chlorid of ammonium in water or a like solution of sulfate of ammonium is used for the electrolyte for either zinc iron or carbon anode. When mercury is the anode a strong solution of cyanid of potassium is preferably used. For the other electrolyte is used the solution containing the gold or silver or both and the gold is deposited upon a copper cathode which has its surface amalgamated.

598,193, Feb. 1, 1898, Emile Andreoli, of London, England.

Relates to the electrodeposition of gold or silver or other metals from cyanid-compound solutions. Peroxidized-lead plates, as anodes and zinc cathodes are preferably used. A cyanid of gold and potassium, or of silver and potassium solution, is not contaminated by metallic salts and remains limpid when peroxidized with lead anodes. The anodes and cathodes are perforated, disposed transversely in a tank, and the gold solution, entering at one end of the electrolyzer, passes through the holes of the electrodes in perfect contact with the surface of the cathodes on which the gold is deposited. The solution flows out through convenient outlets at the other end of the tank.

600,351, March 8, 1898, Edgar Arthur Ashcroft, of Melbourne, Victoria.

Relates to the treatment of ores and products containing copper, iron, and zinc; and refers to patent 546,873, which this process resembles; with the exception that for the iron anodes (either entirely or partially), anodes consisting of the iron, copper-bearing mattes, regulus or crude metal are substituted. The apparatus consists of a mixing tank, consisting of an open tank containing a stirrer, in which the finely-pulverized roasted ores are placed with a leaching solution containing a ferric salt. The leaching is continued for from one-half to two hours, according to the composition of the ore. The greater part of the zinc and a small quantity of any other metals present is extracted. The solution is filtered off, and the residue is fit for smelting as usual. The solution is electrolyzed in divided cells, the diaphragm preventing intermingling of anolyte and catholyte, the zinc solution entering the cathode compartments of a succession of cells, wherein the zinc is deposited; the solution is constantly circulated from the cathode compartments of all the cells, to the anode compartments of the first cell containing a soluble iron anode; from there it goes to the anode compartment of a succeeding cell containing a carbon anode, in which the iron solution is oxidized, so as to render it fit for use again in a subsequent leaching operation.

601,068, March 22, 1898, Frederick W. Whitridge, of New York, N. Y., Administrator of Ernst Werner von Siemens, deceased.

Relates to the method of and apparatus for extracting gold from its ores, residues and tailings. A vessel of any approved form has a grating a short distance above its bottom and a perforated plate resting on the grating. The broken ore is laid on the plate in a thin layer and is covered with a weak solu-

tion of cyanid of potassium of from 0.05 per cent to 0.13 per cent. Perforated air pipes are placed in the bottom of the vessel under the grating. Air or oxygen, or ozone, is forced through the pipes. The liquid is agitated above the ore and the air is forced upward through the ore simultaneously. The ore is treated from four to six times over with the cyanid solution, and each treatment lasts from five to six hours, so that all the gold may be dissolved out of the ore. The gold depositing-vessel is separated into compartments by partitions, each alternate partition being provided with an opening at its lower part. The gold solution is conducted into the vessel through a pipe and is caused to circulate continuously through the compartments by any means, such as a pump. The gold solution is subjected to electrolysis while circulating in the vessel. The anodes are of iron and the cathodes are of lead. The electricity is suplied from 3.5 to 4 volts and a current of from 0.5 to 1.5 amp per square meter of surface of cathodes. The lead cathodes containing from 2 to 12 per cent of gold are removed from the cell periodically and smelted. By repeatedly forcing air through the solution in the vessel and agitating it, the gold can be dissolved by a weaker solution than heretofore used.

603,904. May 10, 1898, Joseph R. Hebaus, of Butte, Mont., assignor of one-half to Frederick P. Conroy, of same place.

Relates to an apparatus for extracting precious metals from ores, tailings and other compounds. A tank has a lining of amalgamated copper which constitutes the cathode. Agitators having arms radially extended, of iron, constitute the anode. By a sun-and-planet motion, the agitators revolve on their axes and travel around the tank to thoroughly stir its contents. In the operation, the tank is charged with water and the agitators set in motion. The ore is pulverized, introduced gradually, and then is gradually added salt, caustic soda, lime, or other alkaline earths. Ores which carry but little silver do not require salt. Ores which carry silver in quantity require 1 lb. of salt to the ounce of silver. To 1 lb. of caustic soda, 16 lb. of lime are used. The charge is now ready to receive the chemical solution, which consists of from one-fourth to three-fourths per cent of cyanid of potassium, and the electric current turned on. Diluted bromin and sodium dioxid are added during the operation, care being taken to have the potassium cyanid in excess and not to add the bromin and sodium dioxid faster than the cyanogen is set free. After treatment of from 2 to 3 hours, the ore and solution are drawn off.

BOOK REVIEWS.

A Text-Book of Inorganic Chemistry. By **G. Senter,** D.Sc., Ph.D. 4½ x 7½ in. (11½ x 18 cm.). 583 pages, 90 illustrations. Price, $1.75 net. New York: D. Van Nostrand Company.

A well-written book, based on the fundamentally correct principle that "a study of the facts on which laws and theories are based should precede the statement of the laws and theories themselves." Under "Steel," p. 538, *wrought* iron is mentioned instead of *cast* iron—an oversight. The book is well printed, on good paper, and is sold at a low price.

Notes on Qualitative Analysis. By **Horace G. Byers,** Professor of Chemistry, University of Washington, and **Henry G. Knight,** Director of Experiment Station, University of Wyoming. 8vo., 181 pages, no illustrations; price $1.50 net. New York; D. Van Nostrand Company.

The authors apologize in the preface for publishing a new text-book on this subject; the reasons they submit are satisfactory to themselves, but not above discussion. The best reason given is that they are confronted with a special environment, and use qualitative analysis principally as an experimental illustration of the chemistry of the metals and to develop laboratory technique. The book is published "chiefly for home consumption." The equations are written in a somewhat in-

tricate manner, which is matched by the complex explanations put forward. Instead of saying, for instance, that Ag Cl is found, by experiment, to be soluble in ammonia, a long explanation is given as to *why* it ionizes and so passes into solution; this, according to the reviewer's viewpoint, is putting hypothesis ahead of observation, and teaching in exactly the reverse of the rational manner.

Handbuch der Materialenkunde für den Maschinebau. By **Dr. Ing. A. Martens.** Part II: Die technisch wichtigen Eigenschaften der Metalle und Legierungen. By **E. Heyn.** Section A: Die wissenschaftlichen Grundlager für das Stadium der Mettalle und Legierungen. Metallographie. Royal octavo, 19 x 27 cm., 506 pages, 489 illustrations, 19 plates. Price, bound, 42 marks (price in New York, $11.20 net). Berlin: Julius Springer.

Another monumental work, such as German skill and patience alone can produce. With something like majesty the treatment moves systematically through the following chapters: General properties of metals and alloys, the phenomena of setting and cooling alloys, cooling curves, Structures of metals and alloys, Methods of investigation, Mechanical properties and hardness, Metals and gases, Shrinkage, blowholes and segregation, Fluidity, Magnetic properties, Electrical conductivity. Each of these chapters is a treatise in itself, systematically divided into A, B, C, etc., subdivided into 1, 2, 3, etc., and these headed *a*, *b*, *y*, etc. The amount of information is so great as to defy abstracting or even cataloging. The manner of treatment, however, deserves noting. It is clear, exact and wherever possible *quantitative*. On the most obscure questions the actual figures are given whenever they exist. The result is highly satisfactory to the searcher for information; he feels that here, at last, are the actual data so far as known. Two rather glaring omissions must be noted: Vapor tensions of metals and alloys; heat conductivity. They are not mentioned. We hope they will be considered in Section B.

Every investigator of metals and alloys who can read German should add this book to his working library.

Elektrometallurgie, von Dr. **Friedr. Regelsberger.** Ten by fifteen centimeters. 140 pages, 16 illustrations. Bound in cloth. Price, 80 pfennigs. (Retail price in New York, 30 cents.) Leipzig: G. J. Göschen.

A carefully written review of the field of electrometallurgy, attractively printed and bound, and sold for nearly nothing. It might be difficult to convince American publishers that the Göschen establishment is not a philanthropic institution; with money to give away.

The writer divides the book into two parts: general electrometallurgy, describing the classification of processes, and special electrometallurgy, describing the details of application to the various metals. The division of the first part is clearcut and practical: (I) electrolytic decomposition of aqueous or similar solutions; (II) electrolysis of melted salts (free from water), or pyroelectrolysis, and (III) pure electrothermal or electric furnace processes, in which electrolysis is absent.

There is very little to criticise in the book; Heroult is not given credit for his work in obtaining aluminium, nor is Gerod mentioned in connection with electric furnace steel. In truth, however, it is not a mere "pot-boiler," but a carefully written book by a competent scientist.

Cours de Métallurgie des Métaux autres que le Fer. Par **Eug. Prost,** Professeur à l'Université de Liége. 88 pages, 483 illustrations. Price, bound in cloth, 30 francs. Paris and Liege: Librarie Polytechnique, Ch. Beranger.

The metals treated are zinc (150 pages), cadmium (6,) lead (126), silver (92), gold (86), copper (178), nickel (33), cobalt (6), tin (40), mercury (24), antimony (18), bismuth (13), aluminum (29), platinum (9), manganese (9), chromium (8), tungsten (7), molybdenum (6), titanium (6), vanadium (10).

Considering that the book is all the work of one man, it is very well-written, American practice being exceptionally well represented for a foreign book, but this is really too large a field to be treated properly in this way. Two hundred and fifty years ago we passed the date at which a well-educated man could know practically all known sciences, and with the advent of the Twentieth Century we have passed the limit at which one man can know all metallurgy. To know well the metallurgy of iron and steel alone, is heavy work for any one man, and for one person to know well the metallurgy of all the non-ferrous metals is a doubtful possibility. As a consequence of these conditions, an expert in each of the metals treated of in this book can find mistakes and omissions, although in the main the information given is complete and reliable. We cannot understand why sodium, calcium and magnesium are omitted entirely.

As a matter of fact, if such a compilation is needed, it should be written in parts, each by an expert in his own line. If Mr. Ingalls were to condense the important principles and facts of the metallurgy of zinc and cadmium into 156 pages, and Prof. Hoffman similarly treat lead in 126 pages, Dr. Peter's copper in 178 pages, etc., a *vade mecum* for the metallurgical student would result which would be as free from errors and as complete as could be produced. But why refer to something non-existent and probably non-obtainable? We have Prof. Prost's book. As the product of one brain and hand it is nearly as good as could be expected; it is more up-to-date than Schnabel's Handbook, but not quite so generally accurate, and it is probable that many people will find it of considerable use and service, particularly metallurgical students.

Traité de Métallographie. Par Félix Robin. 464 pages. 244 illustrations in text, 460 reproductions of microphotographs. Paris: A. Hermann.

Though as a systematized branch of knowledge dating only from 1895, the present status of metallography is indicated by this voluminous treatise, which comprises about 450 large octavo pages, 244 illustrations in the text, and no less than 460 reproductions of microphotographs. While reposing largely on the microscopic examination of polished sections and surfaces after corrosive treatment of various kinds, the science as here presented also relates to physical examination in general, including the investigation of the nature of the electric spark drawn from a specimen, the effect of an electric arc on a polished surface, the determination of thermal and electrical conductivity, and of the magnetic properties.

The heads of the various chapters of the book are as follows: Generalities and methods, preparation of specimens, methods of examination, constituents, diagrams, qualitative analysis, general constitution of metals, study of the principal constituents of alloys, quantitative analysis of steel, copper, lead, zinc, tin, nickel, manganese, aluminium and precious-metal alloys, physical state and disposition of elements, slags, precipitates and powders.

A section of the first chapter has, the author states, the purpose to encourage the general industrial adoption of metallographic analysis, and to this end there is given an account of the material service which the new science can render to a long list of metal industries. A neighboring section gives a sketch of the development of metallography from Sorby, who in 1864 was ridiculed for proposing the microscopic examination of rail sections, to Osmond, whose capital work dates from about 1895. The author omits reference to his own contributions, but in a preface Osmond points out their extent and value, and foot-notes throughout the book give credit and bibliographical references to the work of other laborers in the field.

The treatise is essentially practical in nature, the innumerable methods of examination, analysis, etc., being presented in such form as to give the work the character of a comprehensive handbook.

Industrial Depressions. By Geo. H. Hull. 6 x 9 in., 287 pages; price, $2.75 net. New York: Frederick A. Stokes Company.

An analysis of the causes of such depressions, a classification of the same, an attempt to provide a remedy for such as result from industrial derangements, and a defence of the thesis that "iron is the barometer of trade."

The thesis is not new, but it is brought out in a new way and proved from a new angle. The statement that "overproduction" is a misnomer, and should be called "temporary surplus production," is skillfully defended as an incident of the struggle to bring "regularity of supply out of irregularity of production."

The principal cause of the fluctuations in question, however, is found by the author to be the large volume of *extra* or *optional construction* entered into by the inducement of low prices at the low point of a depression. These contracts bring about a revival of business, an increase in the agencies of production, and then the down grade into the slough of the next depression as soon as these temporary or emergency contracts are filled.

The remedy proposed is for the government to publish monthly all pertinent information in relation to construction under contract and producing power to meet these contracts, so as to forecast accurately the trend of business and the trend of prices.

The analysis is clever, but the remedy proposed appears inadequate. In any case, all persons interested in studying industrial economics will find the book readable, interesting and stimulating to thought.

Handbook for Ironfounders. 3¾ x 6½ inches, 155 pages. Issued by the Frodair Iron & Steel Company, London.

This little handbook deserves notice; it has such an "antique flavour." It contains much reliable information gleaned from experience, and much unreliable information gathered from inexperience and culled from second and third-class books. It is the opposite of clear and scientific; it is usually muddy and scorns exact science. Some of its statements are the opposite of "practical," and others are "gems" in their way. For instance, where is hard-headed common sense to be found in this statement:

> "What the foundryman wants from the cupola at the tap-hole is an iron hot and fluid enough to make a sound casting, regardless of the amount of fuel required to produce it."

Or, consider the mental force behind this:

> "It is not the velocity at which the blast passes into the cupola that drives it to the center, but the force behind the blast."

Or, ponder on the wealth of information in this:

> "Iron cannot be melted hot and fast in a cupola with either too much or too little fuel, and foundrymen have only to melt their iron as hot and fast as it can be melted in a cupola of the size they are using to know that they are not using either too much or too little fuel in melting."

In order to give honor where honor is due, let us say that the above expressions originated with Mr. E. Kirk in his book on "The Cupola Furnace," and are quoted by the technical staff of the Frodair company as nuggets of metallurgical wisdom!

Where the Frodair experts have their own say, however, they have expressed some very commendable ideas. Their hints on cupola practice and blowing machines are good, and the ten pages on the futility of expecting good castings when adding steel to cast iron in the cupola contain very timely and valuable advice. Their data must be taken, however, wherever it professes to be exact, with several grains of salt. *e. g.*, cast iron flowing from a cupola at a "yellow colour" is said to be at 996 deg. C.!

The book smacks of medievalism, and leans toward prejudice instead of scientific progressiveness.